PRACTICAL NOTES

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ON THE

CYANIDE PROCESS.

BY

FRANCIS L. BOSQUI, Ph.B.,

Supt. of the Standard Consolidated Mining Co.'s Cyanide Works, Bodie, Cal.

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CONTENTS.

ILLUSTRATIONS.

PLATES.

DIAGRAMS.

PREFACE.

THE writer has not projected in the following pages an exhaustive treatise on the cyanide process. Such a work will inevitably appear when the practical operation of the process has reached a higher state of perfection, and when its very complex chemistry is more thoroughly understood. At the present time a knowledge of the details of practice in large cyanide mills in South Africa, Australasia, and the United States is somewhat difficult to obtain; the methods of operating, for which each operator claims a certain amount of originality, are, as a rule, secrets jealously guarded, and in consequence we have a rather meager literature on the subject.

Mr. G. A. Packard, in a recent article,* states as a reason for offering only approximate data of cyanide practice in the United States, that nearly all the plants are experimenting, most of them employing experienced chemists, and constantly making improvements, increasing extraction, and decreasing cost of treatment. Mr. Packard would seem to convey the impression that in the United States cyanide practice had hardly emerged from the experimental stage. This may be true of some localities; but at the Mercur district (Utah), the Bodie district (California), and the Cripple Creek district (Colorado) the process has certainly been, for some time, a recognized factor in the treatment of ores and tailings, and has achieved, in the few years of its application, a brilliant and permanent success.

At the same time, to produce a comprehensive treatise on a metallurgical process so comparatively new, and one whose practical development is so rapidly progressing, would obviously be a somewhat difficult task. Moreover, the cyanide process has been applied in so great a variety of ways, to meet so many different

* "Transactions of American Institute of Mining Engineers," vol. xxvi.

conditions, that it is quite impossible to formulate any specific rules of operation which will apply in all cases. Indeed, it would be hard to find any two mills constructed alike, or operated in the same manner.

At the present time, in view of the immense commercial importance of the cyanide process, its chemistry may perhaps be considered the most promising subject for investigation in the whole field of metallurgy, since it is only by means of a better understanding of the chemistry of the process that we can hope to increase its efficiency. This subject, as at present understood, has been only briefly touched upon in these pages. Able researches, however, have been made and published by a number of authorities, notably Professor Christy, Dr. Wells, and Messrs. Eissler, Sulman, Butters, Clennell, Janin, and Smart.

The recent valuable contribution of Professor Christy* opens up a wide field for investigation. The author's own modest words give one an excellent idea of the complexity of the subject. "It became necessary," he writes, "to test *de novo* almost every step in the ground, so that an investigation, which I had hoped to finish in a couple of years, has already taken twice that time, and is now only fairly begun."

After reading this, one feels disposed to accept the rather sinister dictum of Mr. Philip Argall: "It is doubtful," he says, "if the complete chemistry of these changes is fully understood, even by those who have made it a careful study."

Several interesting works have been published on the cyanide process, containing descriptions of large plants in South Africa and elsewhere, with brief accounts of special methods of working. It occurred to the writer, after reading different authorities on the subject, that the published text books were singularly deficient in descriptions of the *technique* of the process—that is, the details of practical operations. What seemed to be wanting was not a *resumé* of the field, which we already have in the excellent work of Dr. Scheidel, but some general system of working which might be modified or altered to suit special cases.

Special methods will continue to be applied to the direct treatment of ores and tailings; but the writer is confident that the *technique* of such methods may be established by making simple

* "The Solution and Precipitation of Cyanide of Gold," in "Transactions of American Institute of Mining Engineers," vol. xxvi., p. 735.

deductions and variations from the system which he has attempted to describe.

Grateful acknowledgments are due to those with whom the writer has been professionally associated in the development of the process, and by whose unstinted aid and advice he has profited; also to the various contributors to the literature of the subject, upon whose writings he has freely drawn in the preparation of these notes.

BODIE, CALIFORNIA.

PRACTICAL NOTES ON THE CYANIDE PROCESS.

CHAPTER I.

HISTORY AND CHEMISTRY.

History.—I shall only briefly allude to the history of the process, which has been given already at some length by Dr. Scheidel* and Louis Janin, Jr.† The main facts are these: It was known as early as 1806 that gold was soluble in a solution of cyanide of potassium. In 1844 L. Elsner published valuable investigations on the reactions of various metals in an aqueous solution of potassium cyanide, establishing the important fact that gold and silver dissolved in such a solution only in the presence of oxygen. His reaction has been expressed in the following equation, generally known as Elsner's Equation:

$$2\ Au + 4\ KCy + O + H_2O = 2\ AuKCy_2 + 2\ KOH.$$

Subsequently Faraday put Elsner's principle to a practical test for the reduction of the thickness of gold films. From this time until cyanide of potassium was metallurgically applied as a gold solvent, a number of laboratory experiments were made, none of which attained to any practical significance. A variety of patents were taken out for processes of recovering gold, silver, and copper by means of cyanide solutions; but it was not until Messrs. MacArthur and Forrest obtained their patents in 1890 that the principle was commercially applied to the extraction of gold and silver. Dr. Scheidel has briefly summarized their patent-claims as follows:

"(1) The application of dilute solutions of cyanide (not exceeding 8 parts cyanogen to 1,000 parts of water).

* "Bulletin No. 5, California State Mining Bureau."
† "The Mineral Industry," vol. i.

"(2) The use of zinc in a fine state of division.

"(3) The preparatory treatment of the ore which has become partially oxidized by exposure to the weather with an alkali, or alkaline earth, for the purpose of neutralizing the salts of iron or other objectionable ingredients formed by partial oxidation."

The MacArthur-Forrest patents covering the use of zinc in a fine state of division, as a means of precipitating gold and silver from a variety of different solutions, including cyanide of potassium, were not obtained until some time later.

Metallurgical experiments on a large scale were made in 1888 on ore from the New Zealand Crown Mine. The commercial importance of the process was at once established; and in 1890 it was introduced into the Transvaal as the "MacArthur and Forrest Process," where it has since been operated with astonishing success.

Chemistry.—This subject has found enthusiastic investigators in all parts of the world; but with the exception of the two main reactions upon which it is generally agreed the cyanide process depends, the matter has hardly advanced beyond an incipient stage of conflicting theories. I have already cited Elsner's equation, which is now commonly accepted as a theoretical explanation of the interesting changes which take place in a pure cyanide solution when brought into contact with gold. By simply substituting silver for gold in this equation we obtain Elsner's reaction for the dissolution of silver.

The reaction which occurs in the precipitation boxes is generally assumed to be one of simple substitution of zinc for gold, thus:

$$2\ KAuCy_2 + Zn = K_2ZnCy_4 + 2\ Au.$$

The correctness of this equation, however, has not yet been demonstrated. Its inadequacy to explain *all* the changes occurring during zinc precipitation has been very ingeniously pointed out by Professor Christy in his recent paper, before cited. There seems to be no doubt that the nature of the changes going on in the zinc-boxes is much more complex than was at first supposed.

The chemistry of the cyanide process is greatly complicated by the active affinity of cyanogen for a variety of other substances beside gold and silver. From all these combinations arise a complexity of chemical reactions which are only vaguely understood. Again, the aurocyanide solutions entering the zinc-boxes charged with precious metals, lime, or caustic soda, and the impurities resulting from the reaction of cyanogen on whatever "cyanicides"

or soluble salts the ore may contain, open up another field of bewildering complications which will probably require many years of investigation to unravel.

"The great complexity of the cyanides," writes Professor Christy, who has perhaps gone further than his predecessors in his investigations of the subject, "the lack of accurate information in the text books, and the consequent erroneous statements in many of the publications on the cyanide process, have greatly added to the difficulty of the study."

Sources of Cyanide Consumption.—We have seen that the secondary reactions occurring between cyanogen and certain soluble salts (such, for instance, as the soluble iron salts resulting from pyritic decomposition) are only imperfectly understood. That these reactions are accompanied by a considerable loss of cyanide is proven by the fact of there being, in all cases, an enormous consumption of the chemical in comparison with what is theoretically required for the actual dissolution of gold and silver. Other sources of cyanide consumption have been pointed out and will be briefly noted.

(1) Practical experience has shown that the presence of certain soluble salts of copper, bismuth, antimony, manganese, arsenic, iron, sulphur, and tellurium have a deleterious action on cyanide solutions, in many cases rendering the process wholly inapplicable.

(2) A very common source of cyanide consumption is sulphuric acid and certain soluble and insoluble iron salts found in oxidized tailings, as the products of the decomposition of iron pyrites. The preliminary treatment of this so-called "acid ore" will be discussed in subsequent pages.

(3) A distinct loss is sometimes traceable to the presence, in old tailings, of organic matter, such as grass-roots, fragments of decayed brushwood, etc.

(4) Some writers have laid considerable stress on the losses occurring from the mere deliquescence of potassium cyanide, and the consequent volatilization of hydrocyanic acid gas. There appears also to be a loss due to the absorption of carbonic acid gas from the air (where the surface of the solution is in contact with the air) displacing cyanogen and liberating hydrocyanic acid gas. A further loss has been traced to the oxidation of cyanide of potassium in the presence of air, and its partial conversion into a cyanate, and then into a carbonate. These changes may be all grouped under the one head of losses due

to contact with the atmosphere. Such losses unquestionably do occur, as laboratory investigations have conclusively shown; but, as we shall find in regard to other points in the process, the results of small tests are not always borne out in practice. All who have had any practical experience with the process on a large scale will be disposed to agree with Dr. Wells,* that "the losses of cyanide caused by air and carbonic acid are, in reality, very insignificant."

Mr. Alfred James† very forcibly refutes the supposition of a possible loss of cyanide from oxidation. "Instead of cyanide of potassium in dilute solutions being easily oxidized into cyanate, as the text books would dispose one to believe, agitation of the solution by compressed air for lengthened periods had practically no effect, and even oxygen passed through a solution for some hours at a time was equally innocuous."

(5) Losses of cyanide by absorption of wooden vats may be appreciable at first, but after a mill has been running for a short time they probably become insignificant. The absorption may depend somewhat upon the character of wood used. A piece of California redwood, exposing 72 sq. in. of surface, immersed for two weeks in 2 liters of 0.2% cyanide solution, produced no noticeable consumption of cyanide, while a piece of white pine, immersed under the same conditions, produced a consumption of 0.03%. Both pieces of wood were thoroughly seasoned.

(6) Theoretically there is a weakening of the cyanide solution in the zinc-boxes by decomposition brought about by the gold-zinc couple; that is, by the electrolytic interaction of the metallic deposit on the zinc, and the unaltered zinc. Sufficient electromotive force is thus established to decompose water, with the liberation of hydrogen. This liberation of hydrogen in the form of small bubbles was first pointed out by Messrs. Butters and Clennell, and is a noticeable feature of all successful zinc precipitation. The hydrogen thus evolved is supposed to carry off, mechanically, a certain amount of hydrocyanic acid gas. Accordingly one might expect a more vigorous electrolytic action from solutions rich in gold, with a correspondingly increased evolution of hydrogen; but a series of laboratory tests made by the writer showed that in a solution containing $25 per ton in gold no more cyanide was consumed in the presence of zinc shavings after an exposure of 15 days than in a solution containing $4 per ton, other conditions

* *Engineering and Mining Journal*, Dec. 14, 1895.
† "Cyanide Practice," "Transactions Institute Mining and Metallurgy," vol. iii., p. 409.

being the same. Both solutions indicated a loss of 0.01% cyanide at the end of 24 hours, and 0.08% at the end of 15 days. It would appear that the loss of cyanide depended primarily upon the length of time of contact of a solution with the zinc. Mr. Packard* is authority for the statement that the cyanide consumption in the zinc-boxes also varies with the strength of solution, and the amount of other salts in solution. "Using a 1% cyanide solution," he writes, "and having mixed with the ore in the tanks an excess of impure lime, containing alumina and magnesia, I have had as high as 3 lb. of cyanide per ton of solution consumed in the zinc-boxes." Ordinarily, however, by comparing the strengths of solutions entering and those leaving the zinc-boxes, no appreciable loss can be discovered. Nevertheless we cannot escape the conclusion that there must be some consumption of cyanide in the precipitation boxes, but only in very small quantity; and that although this consumption is not determinable in any given quantity of solution flowing rapidly through a box, it might be apparent in the whole bulk of stock solution used in 24 hours, if we had any means of estimating it.

(7) Some loss is due to discharging the leached residues saturated with a weak cyanide solution.

(8) A considerable loss is incurred by running to waste superfluous weak solutions.

(9) A variable loss occurs during the first stage of the process, traceable to the action of "cyanicides" or cyanide consumers in the ore, and to the presence of organic matter; also to the dissolution of the gold and silver, as accounted for by Elsner's reaction.

The sources of cyanide consumption at the Standard Works, Bodie, have been tabulated as follows:

	Per day, lb.	Per ton, lb.
Consumption of KCy in spent solution	14.0	0.179
Consumption during actual leaching	15.6	0.200
Probable consumption in zinc-boxes	0.4	0.005
Total consumption of KCy	30.0	0.384

The Necessity of Oxygen.—Another point in the chemistry of the process which has a considerable practical bearing is the necessity of oxygen in the dissolving action of cyanide solutions. The necessity of the presence of oxygen was denied by MacArthur, one

* "Transactions of American Institute of Mining Engineers," vol. xxvi., p. 719.

of the inventors of the process, but was subsequently proven by Maclaurin, the truth of whose conclusions has been confirmed by Christy and others.

The important conclusions reached by Maclaurin are:

"(1) That oxygen is necessary for the solution of gold in potassium cyanide solutions, and that it combines, quantitatively, according to Elsner's reaction.

"(2) That the rate of the solubility of gold in potassium cyanide solutions passes through a maximum in passing from concentrated to dilute solutions.

"(3) That this remarkable variation is explained by the fact, which he also proves, that the solubility of oxygen in cyanide solutions decreases with the concentration of the latter."

In confirmation of these results, Professor Christy says:*

"While Maclaurin's experiments seemed conclusive, still the difference of opinion between Elsner and Maclaurin on the one hand, and the reputed discoverer of the cyanide process on the other, was so fundamental that it seemed worthy of further investigation. The importance of this fundamental reaction is not merely scientific, it is of the greatest practical importance in the application of the process.

"Hence, this was one of the first points to which I turned my attention. The result of my investigation was an entire confirmation of the accuracy of the Elsner reaction. That is, repeated experiments indicated that a solution of pure cyanide of potassium in pure water, from which all other substances are excluded, is entirely without action on metallic gold. Under favorable circumstances such a solution absorbs oxygen from the air (without, as MacArthur assumed, immediately oxidizing the cyanide to cyanate), and the affinity of the potassium for oxygen and water, combined with the affinity of the cyanogen for gold and cyanide of potassium, leads to the formation of caustic potash and potassium aurocyanide, as per Elsner's reaction. When the air present is limited, the reaction stops when the oxygen of the air is exhausted, and begins again when it is supplied."

The discovery of the necessity of oxygen led to the investigation of certain other oxidizing agents, the results of which were a number of published researches, showing that bromine, chlorine, bromocyanogen, peroxide of sodium, etc., when added to a cyanide solution, considerably increased the solubility of the gold, by liberat-

* "Transactions of American Institute of Mining Engineers," vol. xxvi., p. 739.

ing "nascent cyanogen" which directly attacked the metal. In connection with these researches, the point was emphasized that it appeared to be the nascent cyanogen only which acted on the gold.

In actual working, however, the use of such agents has not been found practicable, although they form the basis of a number of special processes, for which letters-patent have been obtained. Professor Christy's conclusions on this point seem to express the gist of the whole matter. He writes:

"On the whole, with low-grade ores and dilute solutions, the cyanide solution itself will, if properly aerated, carry oxygen enough to dissolve the gold; so that artificial oxidizers are seldom needed, unless there is some reducing agent present in the water or the ore which absorbs the oxygen. In such a case, and with richer ores and stronger solutions, there is sometimes a distinct advantage in the use of oxidizing agents. But, unless used with the nicest discrimination, they do more harm than good."

CHAPTER II.

APPLICATION OF THE PROCESS.

It may be stated in general terms that ores containing gold in a fine state of division are suitable for cyanide treatment. If the gold be coarse, a longer time is required to dissolve it; and as time is an important factor in economical cyaniding, it is generally advisable to subject the ore to some method of plate or pan-amalgamation before leaching.

It is obvious that economical results will depend a good deal, too, upon the coarseness or fineness of the material to be leached; if the crushing be too fine or the material contain too large a proportion of slimes, the long time required for percolation will raise the cost of treatment to the extent, perhaps, of making the process commercially impracticable. It is difficult to say just what the coarseness, in terms of screen-mesh, should be. Much will depend upon the value of the material. High-grade ores will bear a longer treatment than others, since they can stand a higher cost of treatment. In general, low-grade material of which only 50% will pass through an 80-mesh screen is amenable to cyaniding by percolation; material of which 70% will pass through an 80-mesh screen (No. 38 American gauge wire) will require suction by some form of vacuum-pump to aid percolation; while the only hope for finer material would be in a combination of prolonged leaching, suction, and the use of a large number of shallow vats, since "the capacity for leaching increases in the direct ratio of the surface areas."

The conditions are so variable, however, that it is perhaps idle, after all, to attempt to prescribe any general rule for fineness. Soft, porous ore, which quickly disintegrates in the presence of solution, like that at the Mercur Mine, Utah, is only crushed to pass a ¾-in.-mesh screen. Much will be found to depend, not alone upon the hardness or softness of the ore, but upon the condition in which the gold is found.

It must be borne in mind that all fine tailings are not such by virtue of the slimes they contain. Their fineness may be due merely to fine crushing, in which case they more readily admit of percolation; it is the presence of slimes, in varying quantities,

which offers the greatest mechanical obstacle to economical cyaniding. In Bodie, tailings of which only 30% were retained on an 80-mesh screen, were leached with great difficulty; on the other hand, tailings from the Ybarra Mine, Lower California, of which as little as 25% were retained by the 80-mesh screen, quite freely admitted of the passage of solutions. The latter contained little or no slimes.

By the term slimes we mean the finely-divided clayey residues which settle, in a tailings reservoir, at points furthest from the tailings-sluice discharge, and pack in the form of dense adhesive layers. When the pond, or reservoir, is finally drained of its clear water, these layers dry and crack up into impervious cakes, which can be readily crushed into the finest powder. If these lumps are mixed in a vat with coarser tailings they are partially disintegrated by the cyanide solution, but only enough to yield up a small percentage of their gold and silver; while they are mechanically objectionable as forming impervious nucleii throughout the mass of tailings, which greatly interfere with the uniformity of percolation. Pulverized slimes, with no admixture of coarser material, are practically impervious to cyanide liquors.

Treatment of Concentrates.—The cyanide process has been applied to the treatment of concentrates in South Africa and elsewhere. Its success appears to depend upon long contact with strong cyanide solutions (0.4 to 0.6%), by reason, possibly, of the coarse character of the gold, and its occlusion in the faces of the pyritic crystals. Agitation, as a means of shortening the time of contact, has been successfully applied.

As yet, the process has not been able to compete to any great extent with cheaper methods in the treatment of concentrates, on account of the length of time required for contact with solutions, and the excessive consumption of cyanide. Very recently, however, a preliminary roasting of sulphurets has been found to facilitate their subsequent treatment by cyanide, and to reduce the consumption of the chemical. It seems probable that with the perfection of some cheap method of preparing this rebellious material, the process will become a serious rival of smelting and chlorination.

Pyritic Ores.—Mr. Louis Janin, Jr.,* reaches the following interesting conclusion after a series of cyanide tests on various pyritic ores:

* "The Mineral Industry," vol. i.

"Many favorable results have been obtained by the experimental treatment of pyritic ores, but it must be remembered that in these tests with large excesses of chemicals (strong solutions of cyanide, etc.) the important factor of cost does not enter. This cost . . . does not arise from mechanical difficulties, but is, owing to chemical troubles, due almost solely to the decomposition of the solution by salts which are always present in partially decomposed pyritic ore. It may be, and probably is, quite possible to neutralize these salts by alkalies; but even then the cost will be high, for the excess of chemicals which must be used has a tendency to increase the consumption of cyanide and of zinc in the precipitating bases."

Since Mr. Janin's article was published, the difficulty with pyritic ore has in some instances been overcome by roasting the material before cyaniding. The indications are that while roasting will not prove a panacea, it may go far toward solving an important problem.

Adaptability to Special Cases.—In certain notable cases, where ores have proven refractory to amalgamation, cyaniding has been applied with marked success. "The refractory character of such ores can be caused by the presence of base metals in combination with sulphur or arsenic, or otherwise by their physical structure, which prevents the gold from coming in contact with the mercury during the amalgamation process." Where gold is found in so fine a state of division that the particles are occluded by films of air, rendering amalgamation impossible, cyaniding might also be satisfactorily applied.

Chemical Limitations.—I have already indicated what may be considered the most serious chemical limitation to the process; namely, the presence of gold and silver in forms in which it is not easily yielded up, and the presence of certain deleterious substances which produce an enormous consumption of cyanide, forming with it compounds useless in the extraction of gold and silver.

Effect of Iron Salts.—Of the deleterious iron salts, those most commonly met with are the soluble ferrous sulphate, $FeSO_4$, and the normal ferric sulphate, $Fe_2(SO_4)_3$. Where these two salts exist together in oxidized pyritic ores and tailings, their reaction with cyanide results in the formation of Prussian blue, if the ferric salt be in excess. "A blue color in the solution or on the surface of the tailings, or in the seams of the staves of the vats, indicates a large consumption and loss of cyanide due to imperfect washing and neutralization of the acidity in the preliminary treatment" (Park).

For the various complex and provisional reactions between the iron salts and cyanogen, I must refer the reader to the articles on the chemistry of the process by Messrs. Butters and Clennell in the *Engineering and Mining Journal*, October 22 and 29, 1892.

For the preliminary treatment of material containing iron salts as products of pyritic decomposition, see Chapter VI.

At the present time, as already explained, the action of cyanide of potassium on the various combinations of the metals as they occur in nature is but little understood. In consequence, we can do hardly more than generalize, in speaking of the chemical limitations of the process. Mr. Janin* obtained a great diversity of results from a series of cyanide tests made on gold and silver ores from widely different localities. He arrives at the following conclusion: "It would seem probable that in ores containing both gold and silver only the oxidized surface ores can be treated with success, both the silver and gold minerals from a depth proving refractory."

This deduction is very interesting, but of course is not intended as an absolute guide in determining the fitness of any ore for cyanide treatment. The only test is an exhaustive analysis of each ore, and a study of the variations it presents in the same locality. After knowing the material itself, the investigator can more intelligently go about his cyanide tests and correct the troubles which arise in connection with dissolving the precious metals and precipitating them.

Copper Compounds.—The presence of copper is popularly considered a most serious drawback to successful cyaniding. In practice, the sulphide, oxide and carbonate ores of copper have been found to cause a varying consumption of cyanide. Authorities, however, differ a good deal as to the extent to which the presence of these ores is prohibitive. It is claimed that the difficulty with copper may be obviated by the use of very dilute solutions, for the reason that such solutions have a more immediate affinity for gold and silver, leaving the copper, as well as other "cyanicides" present, practically unaffected. This view, however, has been recently disputed by good authorities.

"It may be broadly stated," writes Mr. Philip Argall,† "that copper ores are unsuitable for cyanide treatment, even with dilute solutions. I am aware that the statement has been made that

* "The Mineral Industry," vol. i.

† *Engineering and Mining Journal*, Sept. 4, 1897.

dilute solutions have no effect on copper sulphides. Practical experience, however, does not confirm this statement. Copper invariably passes into solution and accumulates until it becomes of such strength as to be precipitated on the zinc in the boxes."

Mr. Argall's opinion on the action of copper sulphides is a confirmation of conclusions reached by Mr. William Skey, of New Zealand, published in the "Report of the New Zealand Department of Mines," 1895. The latter found that virtually all the compounds of copper found in connection with gold are decomposed by a dilute cyanide solution (0.03%).

On the other hand, Dr. Scheidel* claims to have found copper sulphides no impediment to the process. Carbonate of copper, however, "was so readily attacked by cyanide, that its presence proved absolutely prohibitive to the extraction of silver, and interfered seriously with the extraction of gold." In this connection he mentions an interesting experiment made on an ore from old workings in a New Zealand mine.

"This most refractory ore came from old workings in the Sylvia Mine, Tararu, New Zealand, where part of the ledge containing a large percentage of copper pyrites had been exposed for many years to the influence of moisture and the atmosphere; the resulting carbonate was hard, but notwithstanding this its reaction on cyanide solutions was very marked. One and a fourth ounce of such copper ore, finely divided and shaken for less than 15 minutes with a 2.73% cyanide of potassium solution, reduced the strength of the solution to 0.05% of cyanide. The treatment of the ore in question proved that the affinity of cyanide to gold is at least equal to that of cyanide to copper, and very much greater than to silver, as, notwithstanding the rapid consumption of cyanide by the copper compound, upward of 70% of the gold assay-value was extracted by cyanide solution of the usual strength, whereas at the same time absolutely no silver had gone into solution. A preliminary treatment of such ore by sulphuric acid had a beneficial effect on the consumption of cyanide and thereby on the extraction of silver."

There are various accumulations of tailings on old mill sites in Nevada and California, which it might pay well to cyanide if it were not for the presence of deleterious copper salts, derived from the ores themselves and from the copper sulphate used in pan-amalgamation. The partial elimination of copper might be effected by sulphuric acid as suggested by Dr. Scheidel. This preliminary

* "Bulletin No. 5, California State Mining Bureau," p. 16.

treatment, however, would, in most instances, require too much time to be profitable, as the sulphuric acid would have to be completely displaced by water before cyanide solution could be applied. This would hardly pay unless the tailings were of sufficiently high grade.

"When copper compounds exist in a state physically hard, the cyanide solution does not readily act on them; but when the copper compounds are soft, porous and spongy, the action of cyanide is so decided as to interfere materially with its action on gold" (MacArthur).

The amount of copper present will obviously be the question to determine in all cases. The presence of copper may not necessarily render the process inapplicable, unless it is in sufficient quantity to produce a consumption of cyanide too great for economical working.

Antimony.—The same authority as cited above offers some interesting information on antimonial ores.

"In the case of antimonial ores, there is little or no interaction between the antimony and the cyanide, consequently the latter is not taken up; but as gold seems to be very firmly held by antimony, and as the compound is very impervious, the cyanide is unable to penetrate the mass, and to dissolve and separate the precious from the base metals. In the case of both copper and antimony the cyanide solution will act, but in the case of copper, if there is much present and acted upon, the consumption of cyanide is so great that the operation is not profitable, and in the case of the antimonial ores, though the cyanide will act with fine grinding and long contact, the expense involved overbalances the value of the gold contents."

On the other hand, certain ores of the Thames and Reefton gold fields (New Zealand) containing antimonite, the gray sesquisulphide of antimony, could not be treated by the cyanide process, owing to the large consumption of cyanide, and the low extraction (Park).

Silver.—Silver does not yield so readily as gold to cyanide treatment, and for the following reasons: Cyanide of potassium has, chemically, less affinity for silver than for gold; silver rarely occurs in a metallic state in ores subjected to cyanide treatment, and is, therefore, in a measure, occluded from the action of the solution; silver is usually found in combination with bases which readily

consume cyanide; cyanide solutions require long contact with silver to yield good results.

Laboratory tests made by Louis Janin, Jr.,* on the solubility of metallic silver in dilute cyanide solutions, indicated a maximum dissolution of 35.9%, after 96 hours of contact in a 2% solution. The rate of dissolution diminished as the solution was increased in strength up to 15%; beyond this point, concentration produced no noticeable effect on the rate of dissolution.

Certain compounds of silver are more amenable than others to successful cyanide treatment. The interesting investigations of Mr. Louis Janin, Jr.,† on this subject are to the point. He experimented on a variety of silver ores with the following results:

Sample 1.—Ore from Grand Central Mine, Arizona. Silicious, containing large quantities of lime and manganese. Silver minerals, principally cerargyrite and argentite. Never more than 84% recovered by raw pan-amalgamation. Extraction by cyanide, 92.5%.

Sample 2.—Christy Mine, Silver Reef, Utah. Silver found as chloride, sulphide and metallic silver. Traces of carbonate of copper. Yields about 75% by free milling pan-amalgamation process. Extraction by cyanide, 80%.

Sample 3.—Ore from Horn Silver Mine, Utah. Contains large quantities of chloride of silver. Extraction by cyanide, 93.6%.

Sample 4.—Ores from Tybo, Nevada, suffer a severe loss during roasting, with poor subsequent recovery. Mineral is a complex sulphide and fahlore. Extraction by cyanide, 71.8%.

Sample 5.—Ore from Sombretillo, Mexico. Ore almost entirely chloride of silver in a silicious gangue. Extraction by cyanide, 97.3%.

Sample 6.—Ramshorn, Idaho. Ore contains galena and carbonate of lead. Silver associated mainly with the lead mineral. Extraction by cyanide, 80%.

Sample 7.—Rich kaolin ore from Broken Hill deposits in the Barrier Range, New South Wales. Silver occurs in the form of a chloro-bromide, disseminated through the whitish mass. and concentrated on the surface of the quartz and garnet crystals which it contains. A typical, free-milling ore. Yields 95% by amalgamation; 99.7% by cyanide.

Sample 8.—From same mine. Silicious iron ore containing

* *Engineering and Mining Journal*, Dec. 29, 1888.

† "The Mineral Industry," vol. i.

some 38% FeO. Results by cyanide good, but not so satisfactory as in preceding. Extraction by cyanide, 84.6%.

Sample 9.—Tailings from Bullionville, Nevada, resulting from several workings. Contain 10% carbonate of lead, some galena and considerable iron in a silicious gangue. Extraction by cyanide, 32%.

Sample 10.—Bertram & Geddes, Nevada. Silver combined with antimoniate of lead. Ore chloridized in Brückner cylinders, and leached by hyposulphite of soda or lime, with an extraction of 84%. Extraction by cyanide, 11.8%.

Sample 11.—Argenta, Montana. Ore contains over 40% lead, and can only be worked by smelting.

Samples 12 and 13.—Belmont, Nevada, ores. Contain arsenical pyrites, pyrite, blende and galena, with the silver in fahlore, and arsenical and antimonial ruby forms. Extraction poor by cyanide—under 50%.

Sample 14.—Las Yedras, Mexico. Contains large quantities of carbonate of lime, with silver in the form of ruby, silver and arsenical pyrites. Ore worked by chloridizing roasting and subsequent lixiviation. Extraction by cyanide, 41.5%.

Samples 15 and 16.—Ontario & Daly Mines, Utah. Ore contains silver principally in form of fahlore, more or less decomposed. Gives excellent results by amalgamation or lixiviation. Extraction by cyanide, between 72 and 81%.

The Albert Silver Mine, South African Republic, contained 10% of copper, which interfered with extraction of silver. The latter was in the form of a sulphide. Extraction by cyanide, *nil.*

"The conclusions and deductions to be drawn from a study of the foregoing are that silver in oxidized surface ores, or where it occurs as a chloride, is readily attacked by cyanide, and that where no minerals are present which exert an unfavorable influence this method may prove economical. It must be confessed, however, that even with these conditions it has a limited range of usefulness. On the other hand, where lead, oxide of copper, or certain oxides of iron occur, the results are so poor as to preclude the use of the process.

"The results obtained from different samples of silver ore from the same mine vary greatly, for the slight increase of an undesirable element, which would not affect amalgamation in the slightest degree, causes a great decrease in the extraction by cyanide."

However, in certain cases, the process has been successfully applied to silver ores.

"With silver ores, while some very good results have been obtained, the length of time required for treatment has usually been too long, and the consumption of cyanide too high, for the process to give economical results. There are, however, several plants in the vicinity of Tombstone, Arizona, working on silver ores. In the case of ores containing from 1 to 10 oz. of silver, in addition to a commercial gold value, the process has been advantageously employed. Thus the Golden Reward Company, in South Dakota, having certain ores containing from 1 to 5 oz. of silver which was lost in chlorination, has built an addition to the plant, in which such ores are treated with cyanide" (Packard).

In Nevada, silver tailings have been successfully treated by cyanide.

Agitation and Percolation.—The cyanide process, as originally conceived, was a process for treating ores by agitation with cyanide solutions, either in a barrel, as in the barrel-chlorination process, or by means of some form of revolving stirrers. The original plan was to agitate the pulp and solution in small iron or wooden tanks by means of iron blades fixed to a central shaft. After agitating the mass for from 6 to 12 hours the pulp was discharged by means of a stop-cock in the bottom into some form of percolator, or vacuum filter. A suction pump was generally applied to draw off the solution.

As compared with percolation agitation generally yields a higher extraction in considerably less time. On the other hand, it requires motive power, which is in itself a source of considerable expense. The system has been further objected to on the ground of its bringing solutions into contact with iron (an objection which might be eliminated if some mechanical means could be devised for protecting the iron).

An excessive consumption of cyanide in connection with agitation has generally been noted, but has probably been somewhat exaggerated. Other objections are the wear and tear on working parts and the continual watching required.

This system seems particularly adapted to certain hard, high-grade ores, or to small bodies of high-grade concentrates. In practice it has been generally superseded by the percolation method; that is, allowing the cyanide solution to percolate through the mass of ore or tailings, it having been found that although the extraction was more rapid with agitation, the high cost of treat-

ment involved did not warrant the general adoption of this method in practice.

In certain cases, however, agitation has been resorted to where all other methods have seemed to fail. At Angels Camp, California, Dr. Scheidel applied it to the treatment of high-grade sulphuret slimes, obtaining an average extraction of 93%, at a cost of \$3.27 per ton. A description of this plant and the process employed will be found in the chapter on exemplifications of the cyanide process in different localities.

CHAPTER III.

LABORATORY TESTS.

CERTAIN laboratory tests are necessary in order to determine the availability of any given sample of ore or tailings for cyanide treatment. The following points must be investigated:

Extraction:

1. Is percolation possible, or must agitation be resorted to;
2. Approximately, what degree of extraction can be obtained;
3. The loss of cyanide, and the quantity of neutralizing agent necessary;
4. The proper strength for a cyanide solution;
5. Length of time necessary for best economic extraction.

Precipitation:

1. The degree of precipitation from strong solution;
2. The degree of precipitation from weak solution;
3. If precipitation is imperfect, how can it be improved?

It is evident that a very thorough examination is necessary before it can be definitely ascertained whether a given material is economically suitable for cyanide treatment. During the early stages of the development of the process, probably too much stress was laid upon the results of laboratory work; and, in consequence, the application of the process on a larger scale met with considerable disappointment and failure. More recently an attitude quite as unreasonable seems to have been taken; and the opinion popularly prevails that the results of cyanide tests in the laboratory can only be accepted with great reservations. It would be hard to tell whether such results are in general better or worse than those obtained in actual practice. However, it may be safely assumed that if laboratory tests are carried on with every precaution and care, and the same conditions observed as obtain in practice, the results, in the vast majority of instances, may be regarded as trustworthy. Of course, wherever possible, results thus obtained should be confirmed by experiments on 1 or 2-ton lots, or even larger quantities, especially where the erection of large works depends wholly upon the experimenter's report. In some instances com-

plete test-plants, with a capacity of 10 or 20 tons, have been erected at considerable cost, to confirm results obtained on a smaller scale.

Apparatus.—The most commonly used apparatus for making small laboratory tests is the regulation glass percolator, obtainable in varying sizes from dealers in laboratory supplies. But it is very

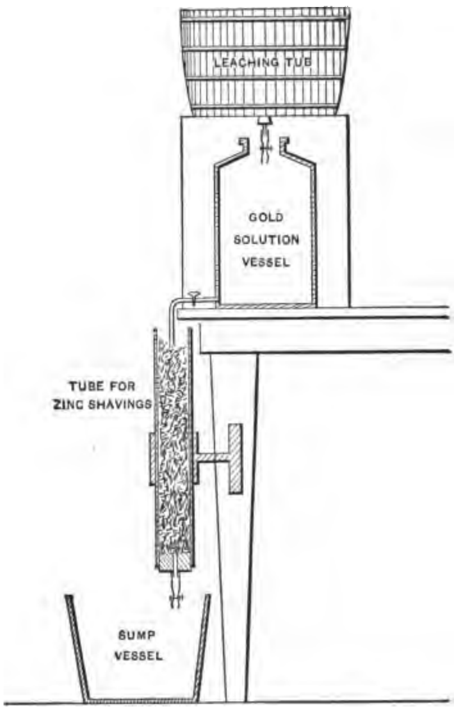

FIG. 1.—EXPERIMENTAL PLANT FOR LABORATORY.

important, as already hinted, that in making such tests the conditions that exist in actual practice should be observed, as far as possible, in the laboratory. For this purpose an excellent leaching and precipitating apparatus may be improvised out of a small tub, a bottle provided with a stop-cock at the bottom for gathering the solution, and a glass tube filled with zinc shavings as an extractor (Fig. 1).

A ten-gallon wine cask cut in two will answer for the tub. The inside should be well coated with paraffine paint to protect the

solution, and a $\frac{1}{2}$-in. hole bored in the bottom for solution discharge. A false bottom may be made by stretching a sheet of thin canvas over a narrow wooden hoop, the hoop being steamed and bent to suit the dimensions of the tub. Two narrow pieces of wood of the same height as the hoop may be interposed between the canvas and the tub bottom, to sustain the weight of material on the filter. A strip of canvas should then be packed down into the space between the hoop and the staves, to keep material from washing under the filter. Into the discharge hole in the bottom is inserted a rubber cork, into which has been tightly fitted a piece of glass tube, the latter being in turn fitted with a piece of short rubber tubing. The flow from the tub is regulated by some form of clamp applied to the rubber tube (the Hoffman adjustable clamp is a good device). The tub is supported on the top of the box, in which a hole has been made to admit the glass and rubber tubing. On the inside of the box is placed the bottle for receiving the solution. Such a tub, with an inside diameter of 10 or 12 in., filled with ore to a depth of 4 or 5 in., would duplicate more closely than a percolator the ratio of height to depth in a large vat. Its comparatively large area of exposed surface of ore makes it better adapted for determining the rate of percolation; and the extraction results are apt to be more reliable than those obtained in a percolator, owing to the greater surface exposed for absorption of oxygen. A bottle or any other suitable receptacle placed under the tub performs about the same function as a gold-tank in a plant.

With each series of tests it is essential that the degree of precipitation of gold from the solution be determined. For this purpose the solution accumulating in the bottle may be allowed to drop slowly into a glass tube closely packed with fine zinc shavings. This tube, $1\frac{1}{2}$ in. in diameter, and about 18 in. long, is supported in a vertical position by means of a clamp attached to the laboratory table. The solution drops to the center of the tube, trickles down through the zinc, and is finally discharged into another vessel (an experimental sump) from which the degree of precipitation may be determined by fire-assay.

The tube is a rather imperfect apparatus for precipitation; but for a small quantity of solution used in a 4 or 5 kg. test it answers the purpose. It may be provided with a cork at the bottom, with glass nipple, tube and clamp, for regulating the flow through the zinc, in cases where the precipitation is slow or imperfect. In

this way solution may be retained any length of time in the tube, to insure longer contact with the zinc shavings.

It is important to determine in each case whether weak solution (about one-fourth the strength of the normal standard solution used) can be precipitated. This is not always possible; and when not, certain variations in the construction and operation of a plant become necessary (described later).

A series of such apparatus as I have described, five or six in number, may be set up along a laboratory table, and will suffice for ordinary tests on tailings. In addition, the following laboratory outfit is required: Six 6-in. glass ignition tubes; one Mohr burette, graduated to 0.1 c.c., with stand; one small wedgewood mortar; two glass pipettes, capacity 10 c.c.; one set small pan scales, with weights ranging from 1 gm. to 500 gm.; one 1,000 c.c. cylindrical glass graduate; one-half dozen evaporating dishes, 5 in. wide at brim; one alcohol lamp; one iron support, with adjustable rings; 1 oz. nitrate of silver; necessary quantity of potassium cyanide, zinc shavings, and lime or caustic soda.

For tests on ore, such an outfit should be supplemented by a set of small screens ranging from 20 to 80-mesh; a mortar for crushing the ore to any fineness; an iron pan with grinder, to revolve by crank, for preliminary amalgamation, and a small oven for roasting, used whenever a preliminary roast of the ore is thought advisable.

Before considering each test separately it will be necessary to describe the best known method for determining the quantity of cyanide in a given solution. There are a variety of methods, but the silver nitrate test is the one generally adopted in practice.

Silver Nitrate Titration Test.—This test depends upon the fact that silver cyanide is soluble in an excess of potassium cyanide, with the formation of a double cyanide of silver and potassium. But at a certain point in the mixture of silver nitrate and potassium cyanide the silver cyanide becomes insoluble and falls as a grayish, flocculent precipitate. The quantity of silver nitrate necessary to be added before this precipitation occurs is recorded on a burette, and from the reading we determine the strength of a given cyanide solution.

A convenient standard silver nitrate solution is one of such a strength that every 0.1 c.c. added to 10 c.c. of cyanide solution, before permanent precipitation takes place, represents 0.01% pure cyanide of potassium. Such a standard solution may be made by

dissolving 13.06 gm. of chemically pure silver nitrate in one liter of distilled water.

A Mohr's burette, graduated to 0.1 c.c., is the most convenient for making this test. The glass float should be removed, the burette filled about two-thirds full of the standard silver solution, and the float replaced. Then with the 10 c.c. pipette a sample of the cyanide solution is taken up and emptied into a test tube (6-in. ignition tubes may be used for this purpose).

From the burette 0.2 or 0.3 c.c. are discharged into the test tube. The tube is immediately shaken to redissolve the precipitate which forms. This process is continued, a drop or two at a time, the subsidence of the float in the burette being carefully noted until a permanent precipitate forms, which does not redissolve upon agitation of the tube. As this permanent precipitate appears somewhat gradually, a mistake may be easily made unless the operator be familiar with the density of precipitate to be obtained. For instance, the faintest permanent precipitate may appear in the test tube when the burette float has subsided 16 points (by points I mean the markings corresponding to 0.1 c.c.), whereas the strength of the solution may in reality be 0.18%. In other words, the precipitate has not been allowed to reach the necessary density. This limit of density is only recognized after some practice, but may be ascertained at once by drawing off 20 points from a burette into 10 c.c. of solution known to contain 0.20% cyanide (2 gm. pure KCy in 1,000 c.c. water). This test tube may be set aside and referred to for comparison until the operator becomes sufficiently skilled in detecting the proper density.

It has been contended that in complex mill solutions containing zinc, the end of the silver-nitrate reaction is indeterminable; that in the presence of zinc the precipitate occurs long before the whole of the cyanide of potassium has been converted into the double salt of silver. Messrs. Butters and Clennell have suggested that "what is most needed is a rapid method of determining the amount of available cyanide for dissolving gold," cyanide in combination with zinc not being available for that purpose. The following method has been suggested: "Two perfectly clean flasks are taken. To each is added 50 c.c. of solution. The liquid in each flask will probably appear slightly turbid, but the degree of turbidity will be the same in each. Standard silver nitrate solution is run into one flask until the slightest possible increase in turbidity is observed in comparison with the liquid in the other flask. This point is taken

as indicating the conversion of the whole of the free potassium cyanide into the soluble silver salt, and therefore determining the amount of available cyanide in the solution."

It is probably only in exceptional cases that the ordinary silver nitrate test will not be found suitable for all practical purposes. In cyanide solutions being continually reinforced by accessions of wash water, the accumulation of zinc is in most instances not sufficient to interfere materially with the validity of the test.

Titrating Complex Solutions. — However, in some cases, as pointed out by J. E. Clennell, "the amount of cyanide in a given solution cannot be accurately determined owing to the presence of various impurities in suspension or solution." He considers the following methods of dealing with such solutions: "(1) Preliminary treatment with lime; (2) titration with iodine; (3) precipitation of the impurities by alkaline sulphides."

The first he condemns because it yielded accurate results "only when the solution contained no soluble double cyanides." "It may only be employed to advantage when the liquid is turbid from the presence of finely divided inert substances in suspension."

The objection to the second was that "solutions containing zinc did not give satisfactory results when titrated with iodine."*

The third he considers the best means of determining the "total cyanide (*i.e.*, the whole of the cyanogen, estimated as its equivalent of potassium cyanide) in solutions commonly met with, particularly such as contain zinc. It depends upon the following facts: (1) That many double cyanides, such as those of zinc, silver, mercury, etc., are decomposed by sulphuretted hydrogen or an alkaline sulphide, with precipitation of the metal as sulphide; (2) that the excess of sulphide may be removed without affecting the cyanides by the addition of insoluble compounds of lead, such as the oxides, carbonate, etc.

The method is as follows: "A measured volume of the solution is made strongly alkaline by the addition of caustic potash or soda. Sulphuretted hydrogen is passed into the liquid until it ceases to give a precipitate, avoiding a large excess, or what is better, a concentrated solution of pure sodium sulphide is added in slight excess. The solution is then well shaken and allowed to stand until the precipitate has subsided. A little lime may be added to assist the settling of the precipitate, in which case it can be filtered

* For method of titrating with iodine, see Park's "Cyanide Process of Gold Extraction," p. 33.

without difficulty. The clear filtrate is freed from excess of sulphide by agitating with litharge, which is best added in small quantities at a time, with constant agitation, until a drop of the liquid no longer gives the slightest black or brown coloration with a drop of lead acetate solution. A definite volume is then filtered off and tested with nitrate of silver in the ordinary manner.

"The liquid to be tested must fulfill the following conditions: (1) It should give a perfectly white precipitate with a drop of lead acetate solution; (2) it should give no precipitate with sodium carbonate; (3) it should give no precipitate with sulphuretted hydrogen. A faint brown coloration is usually produced, probably owing to the solution of a small quantity of lead oxide, or carbonate, in the alkaline fluid.

"In titrating with silver nitrate, a slight granular precipitate was generally observed toward the finish. It was necessary to add the last few drops of silver nitrate slowly with agitation. The end point, however, was perfectly definite, the granular precipitate being disregarded. The point to be noted is the appearance of the distinct permanent turbidity pervading the whole liquid, and not disappearing on standing."

To Determine Amount of Available Cyanide in Commercial Cyanide of Potassium.—When 98% cyanide is not obtainable it is necessary to determine the amount of available cyanide in the material used, in order to make the necessary deductions in preparing solutions. The following test may be made: Dissolve 1 gm. of cyanide to be tested in 100 c.c. distilled water. Titrate this solution with silver nitrate. The number of 0.1 c.c. of silver nitrate solution which it is necessary to use before permanent precipitation takes place indicates the percentage of available cyanide present.

Tests on Tailings.—With tailings the first thing to determine is whether percolation is possible. As a rule, material of which 70 or 80% will pass through an 80-mesh screen is too fine for economical treatment by percolation, unless it is in sufficient quantity and of sufficiently good grade to warrant the construction of a large mill. In such a mill the regulation number of vats would have to be multiplied in order to allow sufficient time for the slow cycle of operations in each vat. Let us assume, for instance, that a plant containing four 75-ton vats will treat in one day 75 tons of tailings of average fineness, and that each vat will require about four days of contact with cyanide solutions. Now, if the material be of such fineness that it requires 12 days to complete the same

cycle of operations, it is obvious that in order to have a capacity of 75 tons a day, this plant will have to be three times larger than the first. It must not be inferred from this that very fine material requires the same time of contact with strong solution as that of medium fineness; the duration of contact must, in each case, depend upon the extraction; but it is the displacing of this strong solution by weak solution and water that prolongs the operation, perhaps beyond economical limits. These are points for which no hard and fast rule can be prescribed, and should be determined by preliminary tests.

In general, tailings through which solution will percolate at the rate of 2 in. or more an hour may be cyanided by percolation; where the leaching rate is only 1 in. per hour some form of suction and a larger vat capacity will be necessary; where the rate of percolation is ½ in. or less per hour, economic percolation is practically impossible, unless the tank capacity is very large or the material rich enough to compensate for the long time of contact required.

The limit finally appears at which percolation must be abandoned in favor of agitation. But as already explained, agitation is only adapted to impermeable material of high grade or to concentrates, where the extraction is sufficiently good to justify the additional expense entailed by use of motive power, etc.

Tests to determine the rate of percolation may be easily made by allowing cyanide solution to percolate through a mass of tailings in the test-tube, and noting the rate of subsidence of the liquid.

A preliminary test may be made to determine roughly, as a sort of starting point for subsequent tests, the degree of extraction. For this purpose mix thoroughly 4 kg. of the dried tailings, take out about 100 gm. for an assay sample, and put the balance into one of the percolation tubs. Level off the charge and pour upon it slowly 4 liters of 0.2% cyanide solution (equal to 8 gm. pure* cyanide of potassium dissolved in 4,000 c.c. water). Shut off the discharge, allow the material to soak 12 hours, then open discharge and drain solution below surface of tailings. About ½ hour after subsidence pour the solution back on top of the charge, and continue this circulation for 12 hours. Then drain and pass through the charge 4 liters of wash water. When the cyanide solution is thus almost completely displaced from the tailings, dry the residues, sample, and assay. The result will ordinarily indicate a low

* By pure is meant the 98% cyanide—pure enough for practical purposes.

extraction, but from this point we may proceed with a series of tests to determine where the fault lies, and apply, if possible, the proper remedy. The percentage of extraction may be determined as follows: Let a equal value of charge per ton; b, value of residues; x, percentage of extraction.

$$\text{Then } x = \frac{a\text{-}b}{a}$$

From the above test gather in separate vessels all the solutions used before wash water is added; and all the dilute solution that runs through after water is added. Let each of these samples pass slowly through the column of zinc shavings, assaying each before and after contact with the zinc. It is advisable to pass the weak solution through much slower than the strong, on account of the less active precipitation from weak solutions.

Method of Assaying Solution Samples.—In assaying the solution a convenient quantity to use as a sample would be 235 c.c. (equal to about 8 assay tons). Put this quantity in an evaporating dish with about 50 gm. of litharge, and evaporate to dryness slowly over an alcohol lamp, or coal-oil stove. Assay the residues. A good flux is 25 parts soda, 50 parts litharge, 2½ parts borax-glass, 22 parts silica, ½ part charcoal.

Amount of Alkali Necessary.—To determine the amount of alkali necessary to neutralize the sulphuric acid and other products of pyritic decomposition in tailings is a matter of first importance. Theoretically, the best way would seem to be to test the sample for acidity. Feldtmann's method, quoted from Park's text book on the cyanide process, is as follows:

(1) Weigh out 224 gm. of ore and shake up with 250 c.c. of water in a tall glass jar or cylinder.

(2) Fill a burette with a standard solution of soda and titrate the ore solution in the jar until the reaction is neutral to test (litmus paper).

(3) Every c.c. of the soda solution used will represent 0.1 lb. of caustic soda to be added to every ton of ore (or tailings) in a wash before the cyanide treatment.

To make standard soda solution, dissolve 10 gm. (or 154.3 grains) of caustic soda in 1,000 c.c. of pure water, and place in a secure bottle.

1 c.c. = 0.1 lb. caustic soda.

The following method of determining soluble acidity is advocated

by Furman.* "Agitate 10 gm. of the pulp for 10 minutes, with 50 c.c. of water; filter, and test the filtrate with litmus paper for acidity. Should acidity be shown, wash the ore until the washings no longer give an acid reaction when tested with litmus paper. Now titrate the total filtrate with decinormal caustic soda solution, until the neutral point is obtained, using litmus as an indicator."

These tests, however, will frequently reveal no acidity, while subsequent operations prove the necessity of using an alkali. This is to be accounted for by what Mr. Furman calls "latent acidity," and for which he offers the following test:

"Transfer the washed ore to a small porcelain evaporating dish; cover with water, add a measured excess of decinormal caustic soda solution; stir and titrate the excess of soda with decinormal acid solution. This gives the latent acidity.

"The sum of the above tests gives the total acidity, but as this is frequently all that is required, it may be determined as follows: Introduce 10 gm. of the pulp into a stoppered bottle with some water; add a measured excess of the caustic soda solution, agitate for 20 minutes, and then titrate back with the decinormal acid solution."

It was found in Bodie practice that the best extraction, and the least consumption of cyanide, was obtained in the presence of a larger quantity of alkali than was called for by laboratory tests for acidity. It would appear then as if an excess of alkali over what was theoretically required exerted some beneficial action on cyanide solutions as yet unexplained. In view of this supposition a more practical procedure would be to make actual tests on samples of tailings in the presence of varying quantities of lime, since the tendency in large plants seems to be to the use of lime in preference to caustic soda.

Let a large sample of the tailings be taken—24 kg., to be more explicit; mix and roll this carefully, and divide it evenly into six lots. With the first lot (4 kg.) intimately mix 2 gm. of pulverized caustic lime (equal to 1 lb. to a ton of tailings); with the second, 4 gm. (equal to 2 lb. to the ton); with the third, 8 gm. (equal to 4 lb. to the ton); with the fourth 12 gm. (equal to 6 lb. to the ton) and so on until the six lots are prepared. Place each lot in a percolating tub, level off the surface of the charge, and intro-

* "Transactions of American Institute of Mining Engineers," vol. xxvi., p. 724.

duce into each tub 4 liters of a 0.2% cyanide solution (8 gm. to 4 liters of water).

Furman calls attention to the fact that the water used in preparing these solutions should be quite pure. "It should always be tested for impurities, for should it contain iron salts, magnesium sulphate, salts with an acid reaction, soluble sulphides, free carbonic acid, or sulphuric acid, they will decompose the potassium cyanide."

The solution is allowed to percolate through the tailings, and when drained below the surface is poured back into the tub again, and so on; this circulation of the solution is continued in each tub for 24 hours. The percolators are then allowed to drain.

Loss of Cyanide.—The loss of cyanide in each mass of solution may then be determined by the silver nitrate test, as previously described.

If in testing the solution from the first tub, the burette float subsides 17 points before the proper density of permanent precipitation occurs, then the solution contains 0.17% cyanide. It has lost 0.03% KCy during percolation, or 0.6 lb. to the ton of tailings. A similar determination may be made for the other five lots of solution, and the one which indicates the least loss of KCy may be taken as the one to which approximately the correct amount of lime has been added.

So much being roughly determined, it is advisable to make another series of similar tests to determine more closely (say to within ½ lb.) how much lime per ton is necessary.

This method may be modified by first making the acidity tests suggested by Furman and Clennell, and determining theoretically the amount of alkali required. This amount may be taken as the starting point for a series of percolation tests, instead of commencing with the arbitrary ratio of 1 lb. of lime to the ton of ore or tailings.

Caustic soda is sometimes used as a neutralizing agent, but in most instances will be found objectionable, from its tendency to foul the solution, and to cause the formation of ferrocyanide of zinc in the precipitation boxes. In Bodie the use of caustic soda was tried and abandoned. When even a very small amount was used it was observed that toward the end of the leaching operations, when the outgoing solutions were well diluted with wash-water, the liquor became so fouled with slimes as to be practically worthless. This may be explained on the theory (not wholly un-

derstood) that the presence of certain salts facilitates filtration by retaining in place the finely divided particles which would otherwise clog the filter and foul the filtrate. Bodie tailings contain a large proportion of slimes. These are kept *in situ* so long as these salts are present in sufficient quantity; but when solutions become dilute, and the soluble salt (such as caustic soda) is washed out, then the slimes begin to clog the filter and to pass out into the solution. It was only when the solution became very dilute that this phenomenon was observed. The same thing was noticed in oxygenation tests carried on with peroxide of sodium.

However, where a large quantity of free acid is present in tailings, and the proportion of slimes is small, a preliminary wash with caustic soda solution, or even with fresh water might be used experimentally; but as such an alkali wash would have to be displaced it might prejudicially prolong the operations.

The prevailing practice, where the tailings are not too acid, is to mix pulverized lime with the ore, as it is being charged into the vats. Lime appears to have no deleterious action on the solution, although an excess is said to increase the consumption of zinc in the precipitation boxes.

The availability of any sample of tailings for cyanide treatment often depends upon the consumption of cyanide; some tests may show an entire consumption of the cyanide present in the solution; others a consumption so great as to render the material wholly unadapted to cyaniding. In each case it is important to determine the cause. With tailings the trouble will, in most cases, be found due to the presence of acid products of oxidation, and may be corrected by using the proper quantity of neutralizing agent. Tailings containing a considerable percentage of copper salts may be considered unsuitable for cyanide treatment. A preliminary acid wash has been suggested and may in some cases be found beneficial.

The presence of large quantities of organic matter may require a preliminary caustic soda wash, or the use of very considerable quantities of lime. Soluble organic matter may be removed by a wash of cold or hot water—preferably hot.

The next point to be determined is the proper strength for cyanide solution. Take 24 kg. of the tailings or ore to be tested; carefully quarter down for an assay sample; mix and roll the remainder thoroughly, and divide into six equal parts. Put samples into six leaching tubs after mixing with each the amount of lime

required (previously determined). Into the first tub introducce 4 liters of 0.1% KCy solution; into the second tub introduce 4 liters of 0.2% KCy solution; and so on for each tub, increasing the strength of solution 0.1% each time, until the six tubs are prepared. Allow each sample to soak 12 hours; then percolate for the same length of time, using the solution over and over again. Drain.

Test the solution accumulating from each tub for loss of cyanide according to method described on a previous page. Then displace the solution from each tub with 4 liters of water, and take sample of drained residues for assay. Compare the percentage of extraction in each case with the respective amount of cyanide consumed. As a rule, a larger proportion of cyanide is consumed from strong solutions. The point to determine is whether the increased extraction is proportionate to the increased consumption of cyanide. (The quotation in New York—October, 1897—on 98% potassium cyanide, was about 28c. per lb.) The proper strength for solution in any particular case is that which gives the best economic extraction; it is obvious that in estimating the best economic extraction many factors must be considered, such as the original value of the material, the cost of cyanide, and the time consumed.

If 0.3% happened to be the most effective strength, similar tests might be made for closer calculations, using solutions of 0.25%, 0.3% and 0.35%, respectively.

Having determined the rate of percolation, the necessary amount of neutralizing agent, and the proper strength for a working solution, we might pass on to an estimate of the length of time necessary for contact with strong solution. This being a final test, at least so far as the first stage in the process is concerned, it will be necessary to observe somewhat more closely the steps which would be followed in actual practice. Take assay sample, as before, from 24 kg.; divide the remainder into six equal parts, and mix each with the necessary amount of lime. Subject each tub to different periods of contact with 4 liters of solution of the proper strength (determined in preceding test); soak contents of each tub 6 hours; treat the first by percolation for 18 hours; the second 30; the third 42; the fourth 54; the fifth 66; the sixth 78 hours. Then instead of displacing the contents of each tub with water, displace with 4 liters of a solution made up to one-fourth the strength of the original strong solution. Allow this to pass through the charge; then wash with 4 liters of water. Drain, and assay the residues.

The results will generally indicate a limit beyond which the

extraction does not increase in proportion to the additional time consumed, and the increased cost of treatment. In some instances the extraction will show no improvement for duration of contact prolonged beyond 48 hours; in others, it will show an improvement proportionate to the time of contact—all depending on the fineness of the gold, whether free or in combination, and on the presence of deleterious substances.

The point to be determined from the previous test is the best economic length of time for contact. For instance, if we assume that the expenses in a 75-ton plant are $75 per day, it is obvious that only a very considerable increase in the extraction would justify prolonging the duration of contact 24 hours. To be more explicit, let us assume that the cost of treatment per hour in a plant treating 2,250 tons per month is $3.10; and that the interest (at 12%) on a plant costing $18,000 would be about $0.24 per hour—making a total cost, per hour, of about $3.34. Now if by prolonging the operations in a 75-ton vat for six hours we increase the extraction 25c. per ton, we really gain nothing, since such an increase would be more than offset by the increase in expense.

Tests on Precipitation.—In many cases difficulties with precipitation have been discovered only after a plant has been constructed and operations commenced. It has often been found impossible to precipitate the precious metals from weak solutions; this is considered one of the most serious drawbacks to the use of zinc as a precipitant. The following points may be determined in the laboratory: (1) The degree of precipitation from strong and weak solutions; (2) if precipitation is imperfect in either case, how can it be improved?

The solution from the previous test may be used in making these important tests. The zinc shavings used should be from the best quality of commercial sheet-zinc, free from antimony and arsenic. Let the strong solution be dropped slowly into the 18-in. zinc-column. Observe the character of the precipitation. Does the zinc darken at the top of the column, or does it become gray at the top, and black toward the end? If it darkens at the top, and remains bright toward the end after considerable contact, the conditions are favorable. When all the solution has passed through, take 235 c.c., evaporate to dryness with 50 gm. of litharge, and assay.

In good precipitation from 85 to 95% of the gold is left on the

zinc; in many instances only a trace of gold can be found in solutions leaving the zinc.

If the precipitation is unsatisfactory in the extractor tube. extend the zinc column by adding another, or even two more tubes. The precipitation from weak solution is usually diffuse, requiring a much longer column of zinc than the strong. If a very poor precipitation from weak solution is shown, even after using three 18-in. columns, increase the strength of the solution up to the point of possible precipitation. Estimate the amount of cyanide required to bring it up to the precipitating point, and determine if this additional consumption of cyanide is justified by the amount of precious metals precipitated. Sometimes the weak solution is sufficiently valuable to warrant the addition of cyanide to bring it up to precipitating strength.

Imperfect deposition of gold from strong solutions may generally be corrected by the use of a longer column of shavings, or by slightly increasing the strength of the cyanide solution.

Results of precipitation obtained in laboratory tests are more apt not to be borne out in practice than extraction results. Consequently such tests should be as exhaustive as possible, and should be repeated on a much larger scale than is practicable in a laboratory. If necessary a small zinc-box might be constructed suitable for treating solutions from 1, 2, or 3-ton tests.

Fouling and Deterioration of Working Solutions.—The clearness of working solutions will be found to vary at different mills. "At the Mercur district, Utah," says Packard, "the presence of a very small quantity of arsenic in the ores soon fouls the solutions so that they cannot be titrated for standardizing; and at most of the mills the solution is thrown away if it becomes badly fouled."

In tailings containing an abundance of soluble iron salts a rusty-colored precipitate appears in the solution, which settles slowly, and interferes with precipitation if allowed to pass into the zinc-boxes. From clean, new tailings containing only a small quantity of oxidation products (as those treated at the Standard Company's Plant No. 1) the solution is perfectly clear and transparent to a depth of 5 or 6 ft. At Plant No. 2, however, where the tailings treated have been accumulating for years, and are considerably fouled by the town drainage, the solution has a bronze coloration, but appears sufficiently clear in a test-tube to be titrated.

Foul solutions may in most instances be titrated after clarifying with lime and filtering.

Theoretically the accumulation of zinc and other substances in working solutions would seem to lessen their efficiency and finally destroy their usefulness altogether. In practice, however, where the working solutions are being gradually renewed by unavoidable dilutions with weak solution and water, these deleterious substances are found not to accumulate to any serious extent. It is generally observed, however, in laboratory tests that a working strong solution is somewhat less efficient than a new solution made up to the same strength.

Tests on Ores.—The direct treatment of ores by the cyanide process has been extensively adopted in New Zealand, and in the States of Colorado and Utah. In some instances it has been found to be the only economic method of treating ores rebellious to ordinary stamp-mill processes. A notable and oft-quoted instance is that the ore at the Mercur district, Utah, "containing gold as a fine coating on particles of magnetic iron in limestone," which yielded only about 40% to amalgamation, and something over 80% to cyaniding.

In making laboratory tests on raw ore, complications arise which do not appear in the case of tailings. It is first necessary to find out the condition in which the gold occurs in the ore; whether in a free state, or in combination with iron, copper, antimony, arsenic, etc.; and whether it be coarse or fine. Ore is only susceptible of profitable cyanide treatment when the gold is in a fine state of division.

Consumption of Cyanide.—A preliminary rough test may be made for comparative purposes, to determine the consumption of cyanide in any given sample of ore. A small sample (about 250 gm.) may be crushed fine enough to pass through an 80-mesh screen, and mixed, in a stoppered bottle, with 250 c.c. of a 0.5% cyanide solution. This may be agitated for 15 minutes or more. The contents are then filtered, and the clear solution tested for loss of cyanide. If the consumption of cyanide seems excessive, a preliminary alkaline wash should be tried, or a direct mixture of lime with the ore, at the rate of 2 or 3 lb. per ton. Obviously a higher consumption of cyanide may be borne with a high-grade ore than with a low. Three pounds per ton might be an excessive consumption with a $10 ore; 4 or 5 lb. with a $20 ore. This will of course depend upon the extraction obtained, and upon many local conditions. What might be considered an excessive consumption in one locality might be considered within economical

limits in another. Much will depend, too, upon the probable cost of cyaniding in comparison with other possible modes of treatment. Where an ore is absolutely unavailable for any other treatment a very considerable loss of the chemical might be borne, providing the process can be applied with some profit.

Preliminary Roasting.—If, when all these conditions are considered, the loss of cyanide is deemed excessive, the cause should be investigated. In many instances pyritic ores, and those containing gold in combination with antimony, tellurium, etc., are benefited by a preliminary roast, the loss of cyanide being reduced, and the extraction proportionately increased.

In the Cripple Creek district, Colorado, a great part of the telluride ore was found absolutely unsuitable for cyaniding without a preliminary roast. After roasting, however, a very high extraction of gold has been obtained, with a normal consumption of cyanide. In some localities certain pyritic ores and concentrates have been successfully treated after a "dead" roast, which could not be treated in the raw state. Roasting accomplishes three purposes, apparently; it drives off the "cyanicides" in the ore, *i. e.*, the volatile substances which consume cyanide, and oxidizes the metallic residue; it may also cause a breaking up of the pyritic particles, exposing more of the gold to contact with solution; in some obscure way it renders the material more readily permeable to cyanide solutions.

In laboratory tests a "dead" roast appears to be necessary to obtain the best results; if the volatilization of the cyanicides is not complete, a considerable consumption of cyanide will be observed.

If the gold in an ore is known to exist in a fine state of division, and comparatively free, and if an excessive consumption of cyanide does not at once disqualify it for treatment, the next point to determine is the extraction. A preliminary test is made by crushing 4 kg. of the sample to 30 or 40-mesh, treating it with a 0.3 or 0.4% solution, following the same procedure as in the test for tailings. If the extraction indicated is low, try finer crushing; if the result is still unsatisfactory, the difficulty may be corrected by roasting, as already explained. If, however, a roast proves of no advantage, the gold may in some instances be liberated, and exposed to the action of solution by very fine crushing—so fine, perhaps, as to make percolation impracticable. Agitation may then be tried; this mode of applying the process has succeeded where all other expedients have failed.

If the results of such tests are in any way encouraging, a series should be made similar to those indicated for tailings. It is important to determine just what degree of fineness insures the best extraction; also the strength of solution and time of contact necessary.

Agitation on Ores.—Agitation tests on ores should be carried on preferably in an open vessel. A tub may be fitted with a central shaft and stirrers, and so placed that it can be operated from a lineshaft. The contents, after agitation, may be discharged into a second tub, provided with a filter bottom, to which suction may be applied by means of a simple hydraulic ejector attached to a faucet discharging a good force of water.

CHAPTER IV.

DESIGN OF WORKS.

In most instances an advantage will be gained, obviously, by constructing the plant on sloping ground. If a flat site is preferred, the leaching vats must be raised on piers of masonry, in order to get the necessary gradient for the flow of solutions by gravitation from one tier of vats to the next below. On flat sites the storage tanks are usually dispensed with altogether, and the solution pumped directly from the sumps to the leaching vats. While the sloping site is in the vast majority of cases to be preferred, the choice will depend a good deal upon the mechanical difficulties in the way of transporting the ore or tailings to the leaching vats.

Where so great a weight of tailings and solution has to be sustained, as in a cyanide plant, the necessity is obvious of selecting only hard, firm ground as a site, and avoiding soft or made ground. It is hardly necessary to say that a plant should be erected as near as possible to the source of material to be treated. A considerable advantage would be gained, in treating tailings, by erecting the plant below the reservoir, so that the vats could be charged and discharged by gravitation. This is rarely possible, however. In any case, the works should be high enough to admit of the discharge of residues by gravitation, and to allow room for the accumulation of these residues, unless there is some watercourse near at hand to carry them away.

The proper site for a plant will in each case depend upon a variety of local conditions, such as the source of water supply, the accessibility of the material to be treated, the proximity of ground for discharging upon, and the means adopted for transporting the ore. Where the country is flat, and water scarce, sluicing out (a cheap and rapid method of getting rid of the spent material) must give place to some other method; either shoveling out into cars through side or bottom discharge gates, or removing the contents of the vats by means of running cranes, as at the Langlaagte Estate Co.'s works at Johannesburg.

Vat Material.—The vats used in cyaniding works have been variously constructed of wood, iron, masonry, and brick, lined with hydraulic cement. Brick and cement vats have been introduced in special cases, where the works are erected for permanent use in the treatment of large accumulations of tailings.

In the Cripple Creek district, Colorado, steel vats were found cheaper than wooden vats, and have been generally introduced. Perhaps their greater durability and their freedom from that much-exaggerated objection to wooden vats, namely, a tendency to absorb cyanide and gold from the solutions, may entitle them to some preference over the latter. Iron, as a vat material, has very recently come into favor in South Africa, where exposure to sun and weather has been found to affect seriously the life and utility of wooden vats.

Mr. Janin says: "Objections to the use of iron tanks are that they are fully as expensive as wooden tanks, are difficult to calk if they leak, and are easily corroded by the solution and the moisture." The comparative price of wooden and iron tanks of the same size is altogether a matter of locality; in Colorado the iron tank is the cheaper. The other two objections, however, are probably common enough. The tendency of wooden tanks to leak is a very much exaggerated objection. Well constructed wooden tanks do not leak, except perhaps a little at first.

Wooden vats have, in general, held the preference, and have been constructed as large as 42 ft. in diameter. The wooden leaching vats in use in South Africa have been described as follows: "These are, in most instances, made circular, that form being the strongest. They are from 20 to 42 ft. in diameter, and from 8 to 14 ft. in height, and should be constructed of well-seasoned lumber, with staves 3 to 4 in. thick, having their inner and outer faces cut to correspond to the arc of circle of the tank, and their edges radial to this circle. The staves are not tongued or grooved, the pressure of the hoops being sufficient, if the tank is well made, to make them perfectly tight. The staves should be at least 1 ft. longer than the inside depth of the tank, and gained from 1½ in. into the bottom timbers, with a chime of several inches. The bottoms are made of 3 by 9-in. deals, tongued and grooved, and put together with white lead, or litharge and glycerine. The hoops should be made of wrought iron rods from ¾ to 1¼ in. in diameter, according to the size of the tank, with threaded ends passing through wrought-iron lugs and tightened by hexagonal nuts.

When the tanks are of large diameter these hoops are made in sections. The outside of the tanks can be painted in lead paint. The bottoms of the vats rest on wooden beams 6 by 9 in. placed 18 in. apart. These beams are placed across the stone foundation, and rest in their turn on planks 1½ by 11 in. The planks are put between the stone foundation and the beams merely to insure a perfectly level surface. It is obvious that tanks holding such enormous weights should rest on good foundations, and in every case where wooden foundations have been used the result has been that the tanks settled, got out of plumb, and leakages occurred."*

Wooden vats may be either square, round, or rectangular. Square and rectangular vats have no advantage over the round; they require more timber in proportion to their capacity, cost quite as much to build, and are more difficult to make tight. In localities where no suitable material is available it will pay to have the round vats constructed elsewhere and shipped to the site of the works, even at a considerable cost, rather than have them constructed of a poor material. The freight charges on round redwood vats shipped from San Francisco to Bodie amounted to one-half as much as the cost of manufacture.

The round vat is to be recommended for its greater strength and durability and its less susceptibility to leakage; while the shape of the square or rectangular vat is perhaps better adapted to the arrangement of cars along one side, where shoveling out over the side is resorted to.

The Best Vat Material.—The proper wood for a good vat material should be one that resists decomposition by moisture, and has no deleterious effect on cyanide solutions. The California redwood (*Sequoia sempervirens*) seems to fulfill these conditions. This wood, beside, is soft and compressible, and well adapted to making tight joints. Oregon fir has recently been introduced into South Africa, and as a vat material seems to have given satisfactory results.

A piece of pine wood immersed in a cyanide solution for 34 hours at the Salisbury Works, Johannesburg, is said to have reduced the solution from 0.3% in strength to 0.05%.

* "The Cyanide Process," Eissler, p. 15. This assertion is rather sweeping. I believe many instances might be cited where wooden foundations have proven perfectly satisfactory. This matter might be more fairly put by saying that where the tanks do not exceed 20 ft. in diameter, the ground is dry, and the works are not designed for long service, the wooden foundation will answer; under other conditions the stone foundation may give better service.

In Bodie a similar test was made with redwood, and the solution was found unaffected at the end of two weeks' submersion. I have never observed any greater consumption of cyanide at the commencement of operations in a new plant than during subsequent operations; although it is said, on good authority, that in South Africa the loss of cyanide from absorption in new plants amounts to 1 lb. per ton of tailings.

Capacity.—It is a matter of some difficulty to assign to any particular plant a corresponding capacity, as it will depend upon the rate of percolation of the material to be treated, the time required for economic extraction, and the facilities for charging and discharging vats. The first plant constructed in Bodie contained vats and tanks of the same number and size as the 75-ton plant below specified, and had an estimated capacity of 100 tons per day. During its four years of operation, however, its monthly tonnage has ranged between 1,000 and 4,000 tons, according to the character of the material.

If we assume an average rate of percolation, with finely divided gold, and good facilities for charging and sluicing out, it may be broadly stated that ores or tailings corresponding to these conditions will require about four days for good economic extraction;* and that a 75-ton plant for treating same will contain vats and tanks† distributed as follows:

Two Storage Tanks (for strong and weak solutions), 12 ft. in diameter and 10 ft. deep, with a capacity of 36 tons of solution.

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* I have already emphasized the fact that no hard and fast rules, either for the construction of a plant or the treatment of ores, can be definitely established. When I say broadly that four days' treatment will usually suffice for satisfactory extraction, I am simply assuming an average for a very wide range, merely for purposes of illustration. M. Eissler, speaking of cyanide practice on the Witwatersrand fields, says: "The time of treatment of each vat varies, and takes from 50 to 140 hours, according to circumstances and the size of the tanks employed." As instances of the wide range of time consumed for treatment at different plants, the following may be cited: At the Central works of the Rand Ore Reduction Co., in the Transvaal, the time consumed for actual cyaniding is 50 hours; at the Crown Reef, about 6 days. The total time consumed at the old Simmer and Jack Works (from the time of starting filling till ready to discharge) was 96 hours; at the Worcester Works the time is 5½ days. At Bodie the time consumed is from 70 to 200 hours. In Colorado from 70 to 100 hours are consumed. The table of size, capacity of tanks, etc., is not supposed to be based upon any arbitrary rule of calculation. It indicates merely what general proportion may be maintained between the total capacity and the dimensions of the tanks, zinc-boxes, etc. For instance, in a 75-ton plant, I have assumed the depth of the leaching vat to be 7 ft. In Colorado and elsewhere, however, these vats are made only 5 ft. deep. The depth of leaching vat will depend upon whether tailings or ores are to be treated, and upon the character of the percolation. Such a table as I have prepared here is necessarily subject to great variation.

† For lack of a better distinction between the much-used words *vat* and *tank*, I have arbitrarily assigned the name *vat* to the receptacles containing the solid material in a plant, as *ore or tailings-vat*, and *tank* to those containing liquid, as *storage or sump-tank*.

Four Tailings or Leaching Vats, 20 ft. in diameter and 7 ft. deep; with a capacity (allowing for solution space on top of charge, and space for filter-bottom) of 75 tons.

Two Gold Tanks (for strong and weak solutions), 12 ft. in diameter and 5 ft. deep, with a capacity of 18 tons of solution.

Two Sump Tanks (for strong and weak solutions) of same size and capacity as gold tanks.

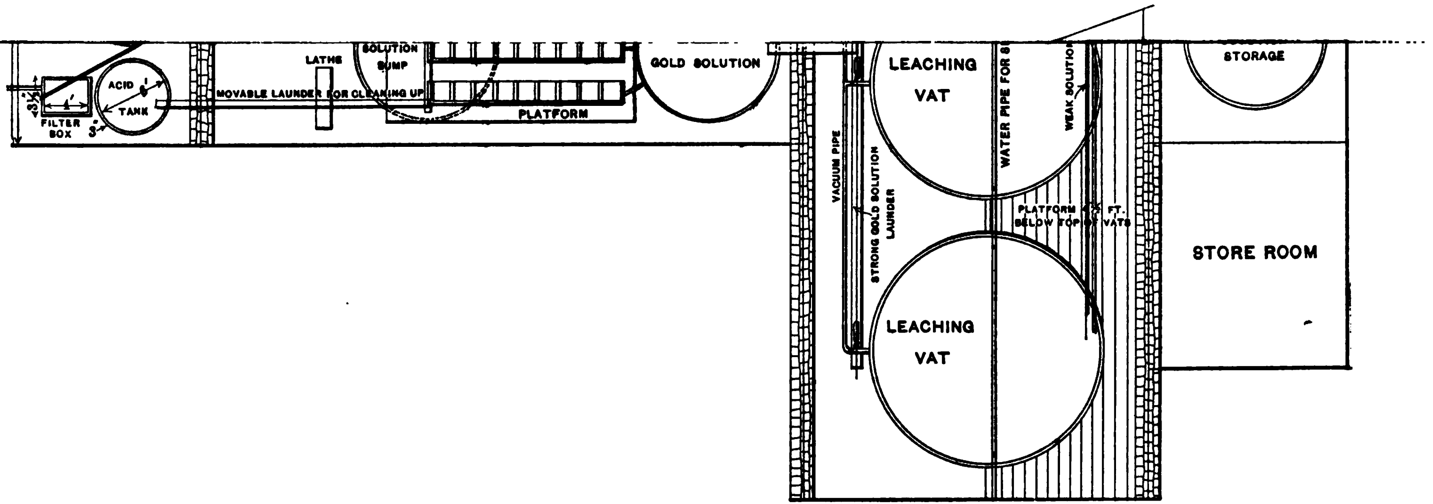
One Acid Tank, 6 ft. in diameter and 2 ft. deep, for the reduction of the zinc-gold slimes.

The following table indicates the approximate number and size of vats, tanks, and zinc-boxes suited to different capacities:

Total Capacity of Plant Per Day.	Storage Tanks.		Ore Vats.			Gold Tanks and Sumps.		Zinc Boxes.			
	No.	Dimensions in Feet.	No.	Dimensions in Feet.	Capacity of Each.	No. of Each	Dimensions in Feet.	No.	Total L'gth in Ft.	No. of Comp'ts.	Size of Comp'ts.
Tons. 75	2	Diam. 12 Depth 10	4	Diam. 20 Depth 7	Tons 75	2	Diam. 12 Depth 5	3 Strong 2 Weak	17	9	24"x18"x15"
100	2	Diam. 12 Depth 12	6	Diam. 20 Depth 7	75	2	Diam. 12 Depth 5	4 Strong 3 Weak	17	9	24"x18"x15"
200	2	Diam. 16 Depth 12	6	Diam. 26 Depth 8	150	2	Diam. 16 Depth 5	4 Strong 3 Weak	17	9	36"x18"x18"
300	2	Diam. 18 Depth 12	6	Diam. 30 Depth 9	220	2	Diam. 18 Depth 5	4 Strong 3 Weak	20	9	36"x24"x20"
400	2	Diam. 20 Depth 12	8	Diam. 30 Depth 9	220	2	Diam. 20 Depth 5	4 Strong 3 Weak	24½	9	42"x30"x20"

As indicated in the table, the storage tanks are not in any case more than 12 ft. in height. An increased capacity is gained by an increase in diameter, a point worth considering when we take into account the height through which it is necessary to raise solutions from the sumps below. The gold tanks and sumps are maintained at a uniform shallowness to save depth in excavating for the different tiers. As to the number of leaching vats, experience has demonstrated that a small number of large vats can be managed more cheaply and with greater facility than a large number of small ones, not to mention the saving in first cost. A diameter of 30 ft. may be assumed to be a reasonable maximum limit in size, although in special instances vats have been constructed much larger than this. Where a greater capacity than 400 tons a day is desired, the number of 220-ton vats may be multiplied, or the size of the vats may be increased. At Cripple Creek, steel vats with a capacity of 500 tons have recently been constructed. In a 400-ton plant, the leaching vats may be arranged in two rows instead of

PLATE I.



Design of a Simple Form of Cyanide Works.
Capacity, 75 tons per day.

one, each row running parallel with the line of incline of the plant. This combination will require some simple modification of the arrangement of pipes and launders, herein described, and also of the method of dumping tailings into the vats.

I have purposely omitted calculations on a capacity less than 75 tons a day. Where a plant is intended to treat only 20 or 30 tons the dimensions of vats and tanks will be in proportion to those given in the table. So small a plant would be adapted only to the treatment of a small quantity of high-grade tailings or to not more than 8,000 or 10,000 tons of tailings of average value. For the treatment of accumulations of 15,000 or 20,000 tons a 75-ton plant would not be too large. The slight excess in first cost would soon be offset by the increased capacity and yield; and while the cost of chemicals would remain about the same per ton as in a small plant, the cost of labor per ton would be less, inasmuch as a 75-ton plant requires no more attention than one of half the capacity. The error is often made of building plants too small. It is obvious, from a financial point of view, that it is better to put up with a very insignificant excess of first cost, if it will insure an output in one year that would otherwise require two.

The necessity for the arrangement of a plant in a series of shelves or terraces, as indicated in Plate I., will be more evident, when we come to consider the matter of operation and management. At present it need only be briefly pointed out that the second shelf supporting the leaching vats should be at a sufficient distance below the first to insure a good fall for solutions in passing from the storage tanks to the body of the tailings; that the vat shelf should be high enough above the zinc-room to allow a good fall for launders discharging from the vats into the gold tanks. There must also be considered the fall required between the gold tanks and the sumps (for the zinc-boxes); and from the zinc-boxes to the acid tank in the clean-up room. Variation in the design of cyanide plants are illustrated in Figs. 2 and 3.

The Building.—In many localities only the zinc-room is covered in. If a plant is constructed for long service, and the leaching vats are built of wood, it is advisable to house in the whole to protect the plant from the weather. If the vats are constructed of cement or iron such precautions are not necessary; in South Africa many of the large plants are, with the exception of the zinc-room, uncovered. In places of high altitude, like Bodie, where the indirect treatment of tailings (that is, by hauling from reservoirs)

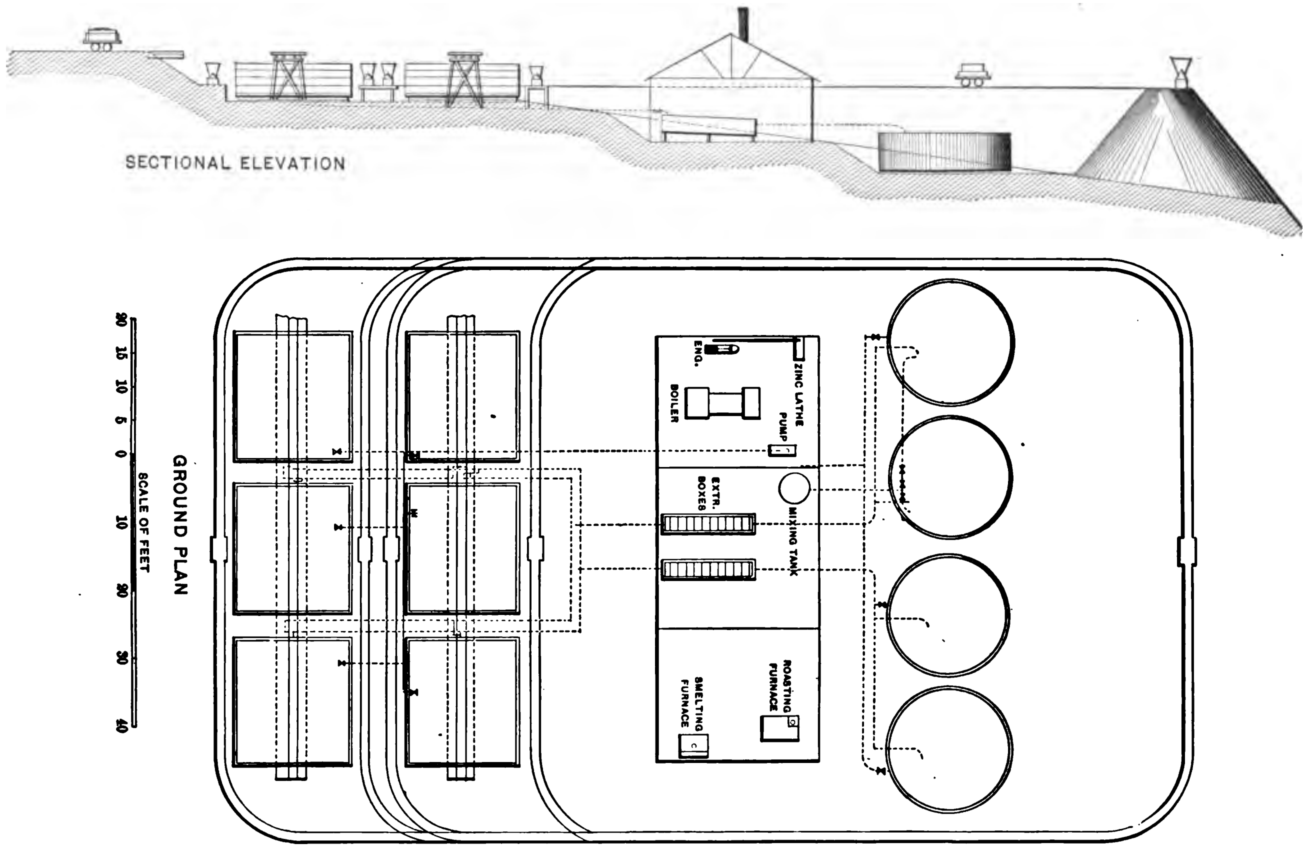


Fig. 2.—Variation in the Design of Cyanide Plants.

Reproduced from Dr. Scheidel's "Cyanide Process," by permission of the Calfornia State Mining Bureau.

is impossible in winter, a sufficient protection was insured in the spring and fall by raising permanent walls on either side of the tier of leaching vats to meet the grating above, upon which the tailings are dumped. Abutting upon this grating shed are two buildings; one above, covering the storage tanks; another below, covering the zinc and clean-up rooms. The character of building, the kind of roofing, etc., are matters to be governed by the conditions in each case.

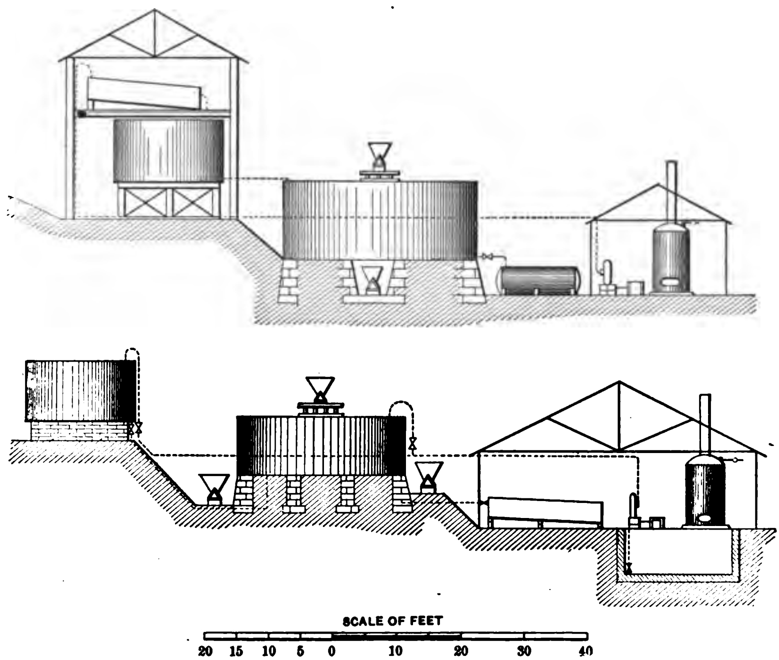


FIG. 3.—VARIATIONS IN THE DESIGN OF CYANIDE PLANTS.

Reproduced from Dr. Scheidel's "Cyanide Process" by permission of the California State Mining Bureau.

Cost of a Plant.—The cost of a plant will obviously vary greatly in different localities, according to the distance from a railroad, the kind of material used in construction, and many other conditions. In South Africa the cost of a plant is given at about 25s. per ton of tailings to be treated per month. "Thus a plant to treat 3,000 tons of tailings per month would cost about £4,000." Such figures, however, must be accepted with considerable margin. The cost in Bodie, owing to the distance from a railroad (40 miles)

and the high cost of labor and material, may be assumed to be about the maximum for Western America. Lumber costs $25 per thousand; laborers receive $3 per day, and carpenters from $4.50 to $6 per day. Specific details of cost of construction, as far as obtainable, are given below.

The Standard plant No. 1 (capacity 75 tons tailings per day) was built when the prevailing knowledge of practical cyaniding and plant construction were comparatively meager. An experience of four years demonstrated the necessity of enlarging the plant, and of improving it in many particulars. At present it consists of two storage tanks, capacity 36 tons solution each; reservoir cut in hillside for water, capacity about 60,000 gal.; four leaching vats, 20 by 7 ft., capacity, 75 tons; two gold solution tanks, 12 by 5 ft., capacity, 18 tons; two sumps, 12 by 5 ft., capacity, 18 tons; three strong solution zinc boxes; four weak solution zinc boxes; a clean-up department, comprising an 8-ton settling tank, a 400-gal. acid tank, a furnace for drying slimes, a water boiler, and a filter box; a boiler and power-room, containing a horizontal locomotive boiler, one Cameron pump, one electric motor (3 horse power), one Gould rotary pump, one 12-in. centrifugal pump, and a circular saw; a grated wooden bridge above leaching vats, over which wagons are driven and dumped.

The cost of the plant as originally constructed, including grading, vats, building, machinery, piping, clean-up apparatus, zinc-boxes, etc., was $16,267.89. The boiler-room and bridge over vats cost $2,202.86, making a total cost of $18,470.75.

The Standard plant No. 2 (capacity 75 tons per day), of the same capacity as No. 1 Plant, differs considerably in the arrangement. Leaching vats are 18 ft. in diameter, and 8 ft. deep; sumps and gold tanks, 15 ft. in diameter, and 5½ ft. deep. There are four strong solution boxes, two weak solution boxes, one Johnson filter press (not included in estimate of cost), one 8-horse power gasoline engine, one Gould vacuum pump (not included in estimate of cost) one 12-in. centrifugal pump, one Gould rotary fire-pump.

This plant cost for lumber and framing in 1895 $1,179.62; for lumber in 1896, $977.16; grading, $1,629.67; roofing, $209.25; nails, iron, etc., $333.34; labor, $1,500; pipe and fitting, $427.11; machinery, $1,119.99; well and reservoir, $398.74; grading roads, $1,061.70; miscellaneous labor, $432.24; sundries, $216.12; vats

at San Francisco, $1,836.16; freight, $935.71; zinc boxes at San Francisco, $320; freight, $40—grand total, $12,616.81.

The Victor plant (capacity 60 tons per day) cost $6,555. It has four 50-ton leaching vats, one storage tank holding 36 tons of solution, one 12-ft. and two 8-ft. gold tanks; two 12-ft. sumps, four zinc-boxes, one Gould rotary pump, one 5-horse power upright engine, one locomotive boiler, four horses and two wagons.

The South End plant (capacity 100 tons per day) cost about $14,000. It has six 75-ton leaching vats, two storage tanks 12 ft. diameter and 12 ft. deep, two gold tanks 6 ft. diameter and 5 ft. deep, two sumps 20 ft. diameter and 7 ft. deep, four zinc-boxes, one lathe for cutting zinc, two Gould rotary suction pumps, melting furnace, assaying room, lime house, one 12-in. centrifugal pump, one 16-ft. horizontal boiler, one Westinghouse, Jr. 25-horse power engine.

CHAPTER V.

DETAILS OF CONSTRUCTION.*

Vats and Tanks.—In view of the great strain to which vats and tanks are subjected in cyaniding works, it is not advisable to build them of staves less than 3 in. in thickness. In some instances the bottom pieces are made thicker than the staves, but this precaution seems superfluous, if the vat is well supported by the foundation timbers. In vats larger than 20 ft. in diameter the staves will have to be correspondingly thicker; in South Africa the largest vats have staves 5 in. thick. The edges of the staves should be beveled slightly and accurately fitted. Dowel pins may be used, and will be found of service in keeping the staves in place while the vat is being erected. A gain of $\frac{3}{4}$ or 1 in. is generally allowed for fitting the bottom pieces into the staves. In no case should the chine be less than 6 in.; if shorter, it is apt to split off when the tank is being set up. The width of the staves will depend upon the diameter of the vat. With a diameter of 20 ft. the maximum width might be about 8 in., although it is not necessary that the staves should all be of uniform width in the same tank.

The bottom pieces may be 12 in. wide and slightly beveled at the ends so that they can be wedged tight into the gain allowed for on the staves. These pieces should be carefully fitted to the staves, as the most obstinate leakages generally occur around the chine. The bottom pieces, in permanent tanks intended for long usage, may be securely nailed to the foundation timbers. A further precaution against leakage may be taken by putting the bottom pieces together with $\frac{1}{2}$-in. dowel strips. If the diameter of the vats exceeds 20 ft. the longest of these pieces might be spliced off

* The reader is referred to Plates I. and V. for a better understanding of the chapters on construction and operation. These drawings are not intended to represent a typical plant, but are merely used to illustrate the practical application of the fundamental principles of the process. Individual and important plants will be described in their proper place.

PLATE II.—EUREKA CYANIDE WORKS, NEVADA.

LEACHING VATS BEFORE ERECTION OF BUILDING.

PRECIPITATION ROOM.

two or more segments, especially if there is any serious obstacle to transporting long timbers over a considerable distance.

Hoops.—The hoops should be carefully made of the best material, as much depends upon their strength and durability. A hoop $\frac{3}{16}$ in. thick and 3 in. wide is suitable for general service, and should be provided with some form of turnbuckle for tightening. Round iron has been used for hoops in South Africa and elsewhere. Flat hoops should be set not further apart than 10 or 12 in., the lowest encircling the line of union between the bottom and the staves. As an additional security an extra hoop may be used near the bottom. Considerable judgment should be exercised in tightening the hoops. If drawn too tight at first they are apt to be broken by the expansion of the vat when liquids are introduced. It is best to draw them moderately while the vat is dry, and then bring them closer, if necessary, when the timbers have thoroughly swelled.

Coating for Vats.—It is always advisable to protect the outside and inside of a vat with some form of preservative coating. The outside may be treated with ordinary lead paint. Perhaps the most desirable material for inside coating is a mixture of asphalt and coal tar, in equal parts, applied hot. A more expensive but less troublesome material for this use is the prepared "P. & B. Paraffine Paint" of commerce. Usually two coats of the latter are necessary to protect the timbers thoroughly.

As soon as a vat or tank is set up and the hoops partially tightened, it is painted on the outside and filled with water. The water is retained until all the serious leakages have "taken up." When drained, the tank should be allowed to dry superficially on the inside (enough for the coating to adhere) before the first coat of asphalt or paraffine paint is applied. By this means the timbers are allowed to expand uniformly, and a certain amount of moisture is retained, which prevents warping or twisting. This method was adopted by the head mechanic of the Standard Works in Bodie, in setting up the 20 or more tanks and vats of the company's tailings plants, and proved to be an excellent one. Through several years of service, leakages have been practically unheard of.

Foundation.—It is very essential that the tanks and leaching vats should stand on a secure foundation. Where the ground is damp, the plant designed for long service, or the vats are unusually large, a stone foundation is to be preferred; but in most instances timber will answer. Those in charge of construction will deter-

mine in each case the arrangement of these timbers. A very serviceable foundation is a series of 6 by 8 timbers, equidistant 2 ft., supported upon a transverse bottom of 3 by 12 mudsills, also 2 ft. apart. The former stand on their narrow side, and are notched at each end to accommodate the chine. The chine should be free, however, the whole vat being supported on its bottom timbers. The ground site should be perfectly level, but the 6 by 8 pieces may be dressed down, so as to give the vat bottom a slight incline (½ in. or more in 20 ft.) toward the solution discharge-valve, to insure complete drainage.

Filter-bottom.—The leaching vats are provided with a false bottom supporting a filter through which pass the gold-bearing solutions. Various forms of filter-bottom have been devised. A layer of sand and gravel covered by a sheet of canvas is used in some large plants; in the majority of cases a bottom of slats or perforated boards is used, supporting cocoa-matting and canvas, or some other durable and pervious material.

At the Standard plants in Bodie a rather expensive but very perfect filter-bottom (a modification of the form originally described by J. S. MacArthur) is in use. It consists of a parallel arrangement of pine slats, 1½ in. by 2 in., standing on their narrow dimension, 1 in. apart; the side abutting on the vat bottom being notched at regular intervals to insure a free circulation of the liquids. The ends of these slats are mortised into an annular ring of wood which extends around the inside of the vat, approaching to within an inch of the staves. This ring is screwed to the vat floor. Between the slats, at intervals, are fixed small blocks to support and strengthen the slats. The slats themselves are fixed firmly to the floor by means of screws. Over the top of this false bottom of slats is stretched a circular, canvas-bound filter of cocoa-matting, which, when tacked down, reaches to the outer edge of the annular ring. On top of this is laid a second filter of 8-oz. duck canvas, made 12 in. wider in diameter than the cocoa-matting. This is stretched into place by means of a 1-in. hemp rope, which is wedged down into the space between the wooden ring and the staves. The advantage of this method of fixing the top filter cloth becomes evident when we consider how often it must be replaced by a new one. A few months of constant wear and tear from the action of sluicing streams, and the shovels and brooms used in removing the last traces of spent tailings, will generally unfit it for further use.

A cheaper filter-bottom, and one quite as efficient, although adapted to a bottom discharge gate for the passage of spent tailings, may be constructed as follows: Slats notched along the bottom (Fig. 4) are laid edgewise in such a manner as to radiate from the center-discharge hole to the staves. These slats are 1½ in. apart at the staves, and 2 in. high; but are dressed down to only 1 in. in height at the discharge collar. This gives an incline to the false bottom of 1 in. toward the center, to facilitate the discharge of water and tailings. The slats do not actually meet the iron collar at the center of the vat, but abut upon an annular ring of wood 2 in. wide, which encircles the collar. To this ring is tacked the filter cloths at the center.

The under filter may be made of the heaviest grade of hop-cloth, which is sufficiently durable, and much cheaper than cocoa-matting. This cloth is tacked down to the stave-end of the slats and

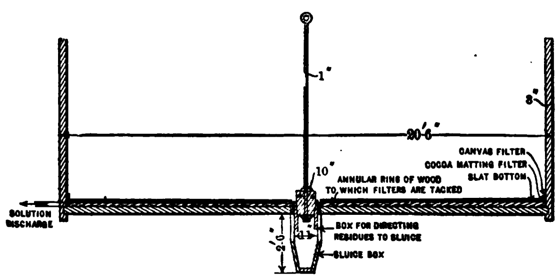


FIG. 4.—DETAILS OF FILTER BOTTOM AND BOTTOM-DISCHARGE DOOR.

to the wooden ring around the center collar. (The hole in the center of the filters may be cut after both filters are laid.) The second filter of 8 oz. duck canvas is made so that when stretched into place it exactly covers the false bottom. If it were tacked down in such a position the tailings would escape around the stave edges and down under the slats. This may be prevented by having stitched around the whole circumference of the top filter, under the hemmed edge, a strip of canvas about 6 in. wide. The filter is then tacked down to the stave-ends of the slats, and the annular strip of canvas brought up and tacked against the staves. This gives a cup-shape to the top filter, and allows nothing to pass under the slats save the solutions. When the two filters are laid the center hole may be cut out along the inner edge of the iron discharge collar.

The heavy under filter is used for the following reasons, which

would seem to justify its use, although it is frequently omitted ; it interposes a good percolating medium between the canvas and the surface of the slats; it prevents wear of the canvas by affording a soft and uniformly level bed; and if there are any unobserved holes in the canvas, it prevents accumulations of tailings between the slats.

In some plants a false bottom of perforated boards is used. In such cases recourse must be had to cocoa-matting as an under filter, to provide a good filtering surface between the perforations.

Discharge Gates for Sluicing.—There is considerable variation in the methods of discharging spent tailings, depending a good

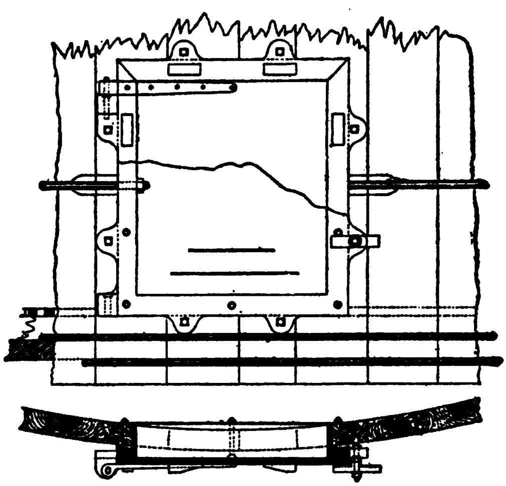


FIG. 5.—FELDTMANN'S SIDE-DISCHARGE DOOR.

deal upon the cost of labor, and the water supply. In some instances in South Africa where water is scarce and native labor cheap, removal of the tailings by shoveling, or by means of running cranes, is resorted to. In general, however, where there is a good head of water, sluicing out the residues will be found to be by far the cheapest and most expeditious method.

A number of side discharge gates have been devised, constructed on a common principle. A frame firmly bolted to the staves supports a hinged door. A rubber jacket is interposed between the frame and the door, and the latter is held tight to the frame by means of a system of bolts, nuts and lugs, or by wedges. At the Standard Company's works two forms of gates are in use, differing

little in principle. One consists of a frame, supporting, by means of two hook-hinges, a movable door, 12 by 18 in., which is held firmly against a rubber jacket on the frame by an adjustable cross-bar and wedge. The other door, which is considerably larger (Fig. 6) is held to the frame by a series of hardwood wedges, driven down behind projecting lugs on the frame. The latter was devised by A. J. McCone, of Virginia City, and was designed originally for the South End Cyanide Company's works at Bodie, where the intention (subsequently changed) was to discharge the residues by shoveling. The gate is hardly adapted to sluicing,

FIG. 6.—McCONE'S SIDE-DISCHARGE GATE.

being unnecessarily large; in fact, side-discharge gates are not, in any case, especially adapted to this method of discharging. It has been found that sluicing may be greatly facilitated if the tailings about to be discharged are first thoroughly saturated with water. This preliminary, if side gates are used, has its disadvantages, for the sluicer rarely escapes a drenching as soon as the great pressure from within is relieved by disturbing the gate. To be sure, he is partly protected by the wooden apron built around the gate, which carries the deluge of water and sand down to the common sluice; but at best the system is a clumsy one.

The bottom discharge is in all respects to be preferred; and I know of none more simple, or more easily manipulated than that recently introduced into Bodie practice by J. F. Parr. It is similar to that devised by W. E. Irvine,* but the former has the merit of being simpler. It consists of a deep flanged iron collar (Fig. 7) which is fitted into a hole in the center of the vat, the collar being fixed to the floor by means of bolts, running through the flange. Into the top of the collar fits a pine-wood plug. The latter is provided with a rod which when adjusted in place projects above the surface of tailings in the vat. In the diagram is represented a collar with a 10-in. opening, adapted to the discharge of a 75-ton vat.

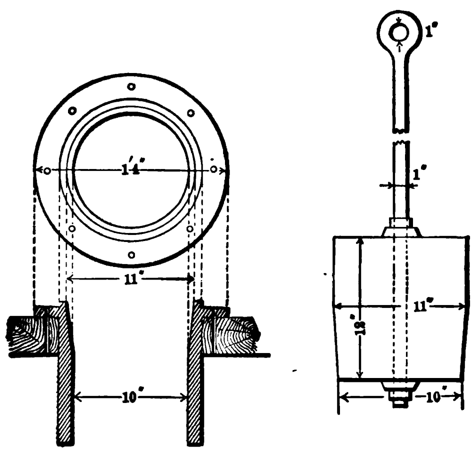


FIG. 7.—PARR'S BOTTOM-DISCHARGE DOOR.

The part of the collar projecting below the vat is inclosed in a box which joins the sluice and prevents unnecessary splashing. The sluice for 2 or 3 ft. on either side of its junction with the box should be covered in for a similar reason.

When adjusting the plug before filling the vat, a few taps with a hammer will suffice to keep it in place. The collar may then be luted around the plug with clay, as a precaution against leakage. At first some difficulty will be had in removing the plug at the time of discharging, owing to expansion of the wood. This may be partly guarded against by giving the plug one or two coats of paraffine paint, and allowing it to dry thoroughly. After one or two trials it will give no further trouble.

* "Cyanide Process," by Dr. E. Scheidel, p. 54.

When the vat is ready for discharging the hose nozzle should be forced into the mass of tailings around the rod to release the pressure on the plug. The latter may then be pried up by means of a crowbar, introduced into the ring at the end of the rod. The plug might be raised by some form of adjustable screw supported across the top of the vat; but such a device would be superfluous, where a simple leverage accomplishes the purpose so well.

A 75-ton vat may be provided with one or more of such discharge holes; the number may be increased in proportion to the size of the vat.

Discharge Gates for Shoveling.—Discharge by shoveling is in use where water is not available, or where, as in the leaching of coarse ore, sluicing is not practicable. At the Robinson Cyanide Works in the Transvaal the tailings are shoveled into tram cars which run underneath the vats between the supporting walls of

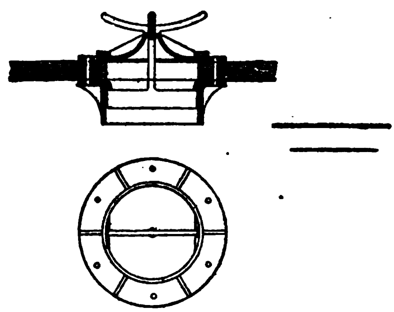


FIG. 8.—BUTTERS' BOTTOM-DISCHARGE DOOR.

masonry. As time is an important factor in discharging a vat, as in all other stages of the process, two side-discharge gates or several bottom discharge doors may be used. A very ingenious bottom-discharge has been devised by Mr. Butters (Fig. 8). The McCone gate is provided with a projecting apron for directing the tailings into the car or sluice; and a sliding door which may be raised or lowered by means of a rope and pulley. Such gates should be placed along a common line, so that a whole tier of vats may be discharged into trucks running on a single track beneath. Plate I. represents a plant in which the tailings are sluiced out into a box running along a stone-walled ditch. A different arrangement would be called for if the tailings had to be shoveled out. A wide, deep trench, lined with masonry walls, would have to be provided under the vats, with a suitable fall for the tracks.

For vats longer than 20 ft. in diameter, shoveling out through side gates would be an awkward matter; in such cases a bottom discharge door, such as that of Butters, which is manipulated from underneath, would be admirably adapted. Several of these gates might be used in a single large vat.

Where shallow vats are used the cars may be run above ground, and the tailings shoveled over the side. In this case it is advisable to have a line of cars on each side to expedite the discharging.

The Sluice Box.—Wherever sluicing out is possible, the sluice box may be designed according to that shown in Fig. 4 with a bottom 2 in. thick and 6 in. wide supporting flaring sides, 2 in. thick and 18 in. high. A box of this shape is to be preferred to a rectangular box, as the narrow bottom gives a greater momentum to the mass of residues and water carried down.

A good fall for a box adapted to carrying material of average fineness would be about 1 in. to the foot; for coarse, heavy tailings a greater fall would be required, 1¼ to 1½ in. to the foot. Coarse tailings have a tendency to bank up in the sluiceway, and require not only a greater fall, but sometimes an additional jet of water in the sluice to carry them along.

Facilities for Transporting and Dumping Tailings.—Economic and expeditious handling of the ore or tailings is of prime importance. Methods vary greatly in different localities. In South Africa steam power is mostly used for conveying tailings to the works.

"The arrangement is simple enough," writes one authority,* "as the dumping cars are pulled up on an inclined trestle work above the leaching vats, and after discharging their contents they run back by gravitation and are held by the brake of the hauling-drum. In large plants, five or six trucks, holding 20 cu. ft., are hauled up at a time. At every mine the mechanical arrangement for filling the tanks is different, depending on local conditions. Messrs. Fraser and Chalmers have recently introduced a system of mechanical haulage by means of endless wire ropes which works very well and which I would recommend in preference to anything I have seen on these fields."

A system of wire-rope haulage was recently introduced at the Harquahala Gold Mining Co.'s cyanide plant in Arizona. The tailings are conveyed to a chute, from which they are transported

* Eissler, "The Cyanide Process," *Transactions Institution of Mining and Metallurgy*, vol. iii.

PLATE III.

GRATED BRIDGE OVER VATS AT STANDARD PLANT NO. 2, BODIE, CAL.

in trucks along the tramways over the vats. It would seem, at first sight, as if such a system, if properly conducted, ought to be the cheapest, providing the capacity of the plant is large enough to admit of its uninterrupted use. It is probable, however, that a rope-haulage system in active service is complicated by a number of petty mechanical problems; and that the wear and tear, breakages, interruptions, etc., involve so much time and attention as to interfere greatly with its economic efficiency.

In Bodie the tailings are transported in wagons by contract at so much per ton. At first they were delivered in 3-ton loads at a chute, from which they were conveyed in cars along a tramway to the vats. This system of double handling was afterward modified and a permanent grated bridge constructed over the vats to accommodate the wagons from which the tailings are directly dumped. This contract system has the one obvious advantage of sparing the management of a plant considerable responsibility and inconvenience. Where tailings are taken directly from the reservoir by cars or bucket conveyors the loading site must be continually changed, necessitating the shifting of tracks, etc., and the expenditure of additional labor and time. However, the cost of labor and the location and character of the material would determine the expediency of any one of the various methods of conveying. It is obviously all a matter of local conditions. In Bodie, where it has been deemed advisable to select only the coarser tailings, and to leave the slimes for subsequent treatment, and where an admixture of the coarser and finer elements is resorted to, as far as practicable, it is found that hauling by wagons is very well adapted to this almost daily shifting of the site of loading.

The following table gives details of transporting tailings in different works in Bodie and vicinity:

Plant.	Method of Conveying.	Time Required to fill vat.	Capacity of Vat.	Distance Covered.	No. of Wagons.	No. of Horses.	No. of Drivers.	No. of Shovelers.
Standard No. 1, Bodie..	By wagons.	8 Hours.	75 Tons.	1,500 Ft.	2	8	2	3
Standard No. 2, Bodie..	"	10 "	75 "	2,000 "	2	6	2	2
South End, Bodie.......	"	4 "	75 "	2,500 "	4	10	4	4
Victor, Bodie............	"	10 "	50 "	1,500 "	2	4	1	2
Eureka Works, Nev....	Endl'ss Belt Conveyor.	7 "	125 "	150 "				

In Bodie the grating-floor through which the tailings fall into the vat below is constructed of 3 by 6 in. scantling, placed 2 in. apart. This floor is in turn laid on a longitudinal series of 6 by 12 in. timbers, standing on edge, 2 ft. apart. The whole is supported between the vats on bents constructed of 10 by 10 in. timbers, the end posts being mortised into a 12 by 12 in. sill, sunk in the ground. Over the center of the vats the grating is swung in a stirrup of 10 by 10 in. timber, suspended by means of 1-in. rods to the roof-trusses of the building. The span over the vats may be further strengthened by a series of cables or hog-chains stretched under the stirrups, and over the bents, and fixed outside the building to suitable anchor posts.

It may be noted that no part of this system is supported by the vats themselves. It is not improbable that the jarring of passing teams might considerably weaken the vats, or interfere with percolation by causing the tailings to pack.

Such a system, applied to tailings, is to be preferred to direct dumping from a tramway. Beside preventing packing, it insures a uniform density throughout the mass, and sifts out lumps of slimes and whatever bulky foreign matter may have gotten mixed with the tailings. The tramway system, however, appears to be the best means adapted to conveying dry-crushed ores from the ore-bin to the vats.

The Precipitation Department.—This contains the gold tanks, zinc-boxes and sumps, beside the pump used to raise solutions from the sumps to the storage tanks, and the suction-pump for hastening percolation. Whether this room contains the source of power for the two pumps will depend upon what kind of power is available. If steam is used, perhaps a separate room for boiler and engine might be constructed on the next shelf below, adjoining the clean-up room. Where the use of power is intermittent, and for short periods of time, as in a cyanide mill, the advantage is obvious of having the power near at hand.

An electric motor or gasoline engine might be conveniently set up in the zinc-room, near the pumps. Electric power is admirably suited to the kind of work required; and where, as in Standard Plant No. 1, it demands only a small fraction of the power furnished to neighboring works and mines, it is unquestionably the cheapest. Gasoline furnishes an excellent power, and is, in most instances, cheaper than steam.

It is well adapted to the intermittent character of the work

required, there being absolutely no waste of fuel in the intervals between pumping. At Standard Plant No. 2 an 8-horse power gasoline engine (manufactured by Union Gas Engine Co., S. F.), has been in successful operation since the erection of the plant. It operates, from a countershaft, a 12-in. centrifugal pump, which raises solution from sumps to storage tanks; a Gould vacuum pump and a Johnson filter press for filtering the zinc slimes. The same engine also operates, by means of a 150-ft. power-rope, a Gould Rotary Fire pump, which raises water from a reservoir below the plant through a 3-in. pipe to a height of 87 ft. This water, pumped up to an 8,000 gal. tank at the rate of 6,000 gal. per hour is used for sluicing, wash-water, etc.

The Gold Tanks.—These tanks stand on a raised foundation, but are placed low enough to receive the solutions from the launders above, and high enough to discharge by gravitation into the

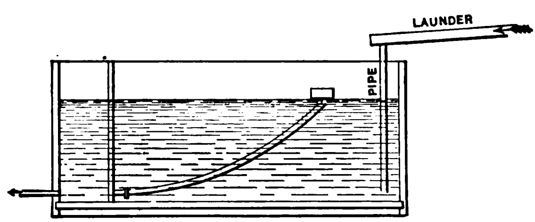


FIG. 9.—FLOATING HOSE FOR GOLD TANK.

zinc-boxes. They serve a double purpose. They accumulate the gold-bearing solutions from the leaching vats, and so provide a uniform discharge into the zinc-boxes; and also act as settlers and filters for solutions containing aluminous or ferrous precipitates thrown down during percolation. These exceedingly light, flocculent precipitates, if allowed to enter the zinc-boxes, very seriously interfere with the deposition of gold on the zinc.

Floating Hose for Gold T nks.—They may be in most instances, very effectually retained in the gold tanks by means of the floating hose device shown in Fig. 9. This is so constructed that only the superficial and clearer solution passes to the other side of the water-tight partition, and so into the zinc-boxes. One end of a 2-in., 5-ply rubber hose is fixed to the bottom of the partition by means of a nipple, sleeve and bushing; and the other end is suspended to some suitable float. The solution entering the tank is carried in a perpendicular box to within a few inches of the bot-

tom, at a point furthest from the partition. Thus currents throughout the mass of the liquid are as far as possible prevented, and the precipitate settled and retained in the tank. At intervals this tank may be cleaned out through a suitable pipe or hose, and the precipitates, if of sufficient value, saved.

This device is a modification of that used in the Russell process for separating the hyposulphite solution from the precipitated silver salts.

In some instances, however, the floating hose seems ineffectual to clarify the solution which passes foul into the zinc-boxes, in spite of every precaution. When this occurs boxes containing supplementary filtering compartments may be arranged at the head of the zinc-boxes. These compartments are of the same dimensions as those in the zinc-boxes, and are filled with oakum, which appears to be an excellent filtering material. Each compartment may be provided with a top screen properly retained in place to keep the oakum from floating off. The oakum may be washed, and the compartments cleaned out once or twice a month.

Canton flannel bags attached to the valves discharging into the zinc-boxes have been used as filters; but the finely-divided precipitate soon clogs them and they are quickly worn out by the pressure to which they are constantly subjected. Besides, filters of this sort require constant changing and washing.

Zinc or Precipitation Boxes.—In the precipitation boxes, the precious metals are recovered from the cyanide solutions. In the majority of instances wooden boxes have been used, although iron has proved an excellent material. Dr. Scheidel states that he could detect no loss of cyanide in iron boxes due to galvanic action of a possible zinc-iron couple; and considers that this phenomenon, as a source of cyanide consumption, has been greatly exaggerated. A similar and apparently unwarranted objection has been urged against the use of iron screens in zinc-boxes. If such screens are properly protected (as they should be, in any case, to preserve the iron) there can be no objection to their use.

For long service iron boxes are perhaps to be preferred, on account of their greater durability, and less liability to leakage. Wooden boxes are very difficult to make tight; and even when apparently perfect at first, are apt to split and crack with prolonged use. It is probable that when certain theoretical prejudices against iron boxes shall have disappeared, they will supersede wooden boxes in all cyanide plants designed for permanent service.

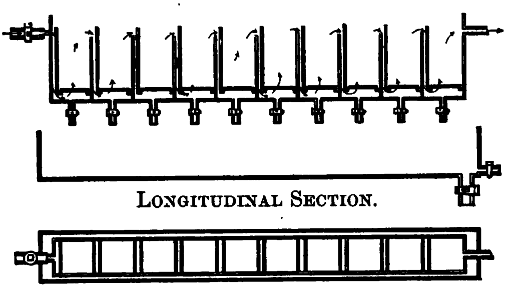
PLATE IV.

ZINC PRECIPITATION ROOM, STANDARD PLANT NO. 2, BODIE, CAL.

The strong-solution boxes are shown in the foreground, the weak-solution boxes in the rear The weak-solution gold-tank stands outside of the building.

Pine is to be preferred to redwood as a zinc-box material. Redwood boxes introduced into Bodie practice had an obstinate tendency to leak around all the lower screw and bolt-heads—owing probably to the softness of the wood. The tendency of pine boxes to split along the sides and bottom after being used awhile may be minimized by selecting only the best grade of seasoned lumber, and by observing the same precautions in protecting the wood with paint as were suggested in connection with tanks. When the box has been set up and painted it should be kept filled with water for several days before being used for cyanide solutions.

Construction of Zinc-Boxes.—The precipitation box (Plate V.) is so constructed that solution entering from the gold tanks at *a* rises at *b* through the screen *c* which supports the zinc shavings. It overflows between the two partitions *d* and *e* and rises again at *f*. The advantage of this arrangement will be considered in the sec-



LONGITUDINAL SECTION.

HORIZONTAL SECTION.

FIG. 10.—SCHEIDEL'S STEEL PRECIPITATION BOX.

tion on operations. The partition *d* is mortised into the bottom and sides of the box, and rises to within 1 in. of the top; partition *e* begins 2½ in. from the bottom and rises flush with the top.

It is evident that the purpose of this mode of circulation of the solution—upward through the shavings—will be defeated if solution is permitted to leak from one compartment into the next, around the edges of the partition. It is therefore a matter of first importance, in constructing a box, to see that each compartment is water-tight.

Screens.—The screen supporting the zinc shavings may be made of No. 14 iron wire, ⅛ in. mesh. The sides of the screen should be bound to a depth of ½ in. with Russia iron strips, and provided on two sides with iron handles, by means of which the screen and

shavings may be lifted. The screen rests on ½-in. cleats fastened to the partitions.

The bottom of the zinc-box is mortised into the sides. The whole should be put together with screws, and further strengthened by bolts passing through the space between the compartments and held on the outside by iron strips. The box is constructed with a narrow false compartment at each end; that at the head, 3 in. wide, receives the solution from the gold tank; the one at the tail of the box, 2 in. wide, receives the solution from the last zinc compartment before it discharges into the sump. The joints in a zinc-box may be made tighter by giving all the mortise-grooves a heavy coat of paraffine paint before putting the parts together.

A variety of means have been suggested for emptying the extractor boxes of their accumulated gold-slimes at the clean-up. It is advisable to have each compartment discharge into a common launder, so that all the material may pass by gravitation from the boxes into the proper tank in the clean-up room. All handling of the slimes should be avoided as much as possible; the system of discharging the slimes into buckets by means of a stop-cock, or of siphoning off the zinc-box solutions, is especially to be condemned.

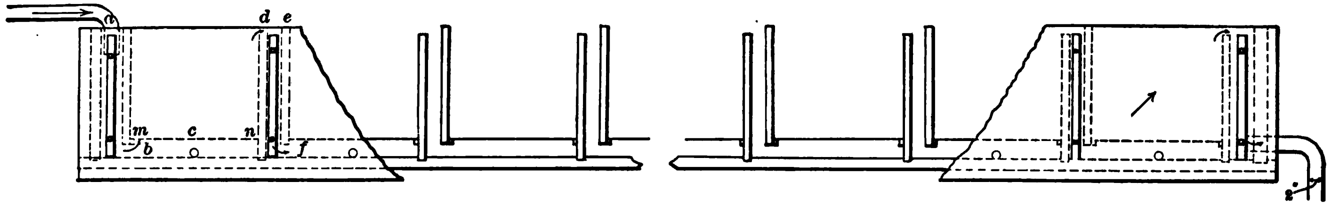
The Side Launder.—An excellent arrangement, and one quite commonly used, is that of a permanent discharge launder fixed to the side of the box. The box, as shown in the diagram, is provided with an inclined bottom, down which the slimes may be swept into the clean-up launder. Each compartment should have a 1-in. discharge-hole, close to the bottom, which may be plugged with a No. 0 rubber cork.

Each clean-up launder discharges into a cross launder, which conveys the material into the clean-up room. This side launder attached to the boxes should be provided with a hinged cover and padlock, to prevent any tampering with the corks. The zinc-boxes themselves are also provided with covers, either of wood or of iron grating. Through the grating the operations in the box, the condition of the zinc, etc., may be observed without the inconvenience of raising the cover, except when the zinc is to be shifted.

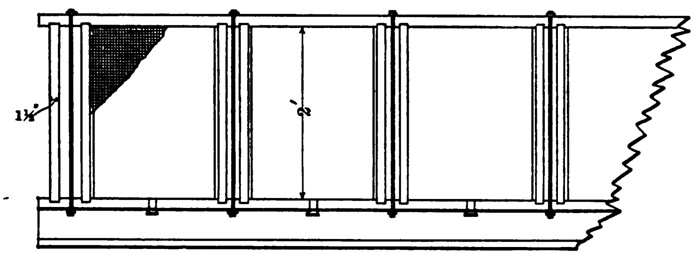
Three such boxes as I have described, containing nine zinc compartments each, will accommodate all the strong solution in a 75-ton plant; and two more, similarly constructed, all the weak. Extremely long boxes, such as were at first introduced, have given place to shorter boxes, 15 to 20 ft. in length. It is as well to mention, however, that the length will be governed a good deal by the

PLATE V.

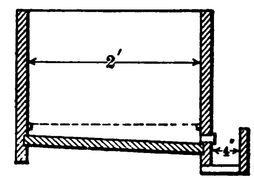
ELEVATION.



PLAN.



CROSS SECTION.



ZINC PRECIPITATION BOXES.

character of the precipitation, a point to be determined by laboratory tests. In base tailings, or those containing much organic matter, the precipitation is apt to be diffused and the precipitate of variable color—being gray in the first compartments and blacker and richer toward the end. Such tailings will require long extractor-boxes. With clean tailings, however, comparatively free from base matter, and carrying an average percentage of gold, the precipitation is very active in the first compartments. At a small 20-ton plant in Bodie, operating for several months on high-grade gold tailings, the precipitation was so complete in the first three compartments that it was found unnecessary to supply zinc to the rest of the box.

But in most instances the dimensions above prescribed will be found to answer the purpose. There is some advantage in a short, broad box over a long, narrow one; the shorter the box the less space required for a zinc-room—the width of the boxes making little difference, as the room must originally be wide enough to accommodate the two gold tanks. The depth of the zinc compartment is purposely made the shortest dimension to facilitate cleaning up.

A box 18 ft. long should have a total inclination of about 4 in. A greater inclination than this would not only be superfluous but detrimental, as any increase in inclination decreases the liquid and zinc capacity of each compartment. In some plants boxes are constructed with the partitions standing at a considerable angle with the bottom, so that when the box is set up at an inclination of 6 or 7 in., the compartments are perpendicular with the floor of the room. This no doubt improves the looks of the box, but in point of utility is quite unnecessary.

In South Africa and New Zealand it was at one time customary to have but two zinc-boxes in a plant—one for strong and one for weak solution. One great advantage in having several boxes is that the discharge from the gold tank may be distributed into slower streams and better precipitation insured; also that operations need not be suspended during the clean-up, or when the flow through any single box is, for any reason, interrupted.

Raised Floor Under Zinc-Boxes.—The zinc-box should stand at a convenient height from the floor. The boarded floor (see Plate I.) may extend for a considerable distance over the sump tanks, and have an inclination sufficient to drain off solutions overflowing from the zinc-boxes. It ought to be covered with some

impermeable material, such as linoleum or paraffine roofing. The space between the boxes, which should be 2 ft. or more, may be provided with a grating to walk upon. After a clean-up this grating may be raised and the floor underneath thoroughly sponged to take up any gold slimes that may have escaped during the process of cleaning up.

The Clean-up Room.—A well-equipped cyanide mill should have a separate room for the reduction of the zinc-gold slimes taken from the zinc-boxes. Such a room should be provided with a concrete floor, to insure the easy recovery of any particles of slimes lost in the handling; and an equipment for treating the slimes.

I have discussed in another chapter the advantages of each method of treating the zinc slimes (by acid and by roasting.) The apparatus used for roasting is much simpler than the acid plant. The roasting apparatus introduced by Chas. Butters at the Rand Central Works consists of a "muffle-roasting furnace in which to dry and roast the slimes. The bottom of the furnace consists of a cast-iron pan, and the wet slimes are charged on to this pan, and, when dry, a damper is closed, which turns the flame through an opening in the fire-bridge, under the iron pan, and the slimes are carefully stirred to avoid dusting, and as much of the zinc as possible is driven off during the roasting" (Eissler). "In New Zealand cyanide works the roasting furnace consists of a large flat cast-iron plate, with raised edges. It is built over a small grate or furnace, and a hood of light sheet-iron is placed over the roasting place, so as to carry off the zinc fumes" (Park). For method of roasting slimes see a later chapter.

If the acid treatment is preferred, the equipment should consist of an acid tank, 2 ft. deep and 6 ft. in diameter, for reducing the zinc with sulphuric acid; a filter-box for filtering the refined precipitate; a settling tank, suitable for holding the liquor drained off from the acid tank in the process of washing; a furnace for drying the precipitate; and, if conversion of the precipitate into bullion is not carried on elsewhere, facilities for melting. If both drying and melting are carried on in the clean-up room, a furnace may be set up with some form of oven annexed for drying.

Acid Tank.—The tank for the reduction of the zinc slimes with acid should be placed low enough to receive the discharge from the zinc-box launder, and high enough for its liquors to be siphoned off into the settling tank, and its solid material to be discharged into the filter-box. It may be constructed of redwood (which

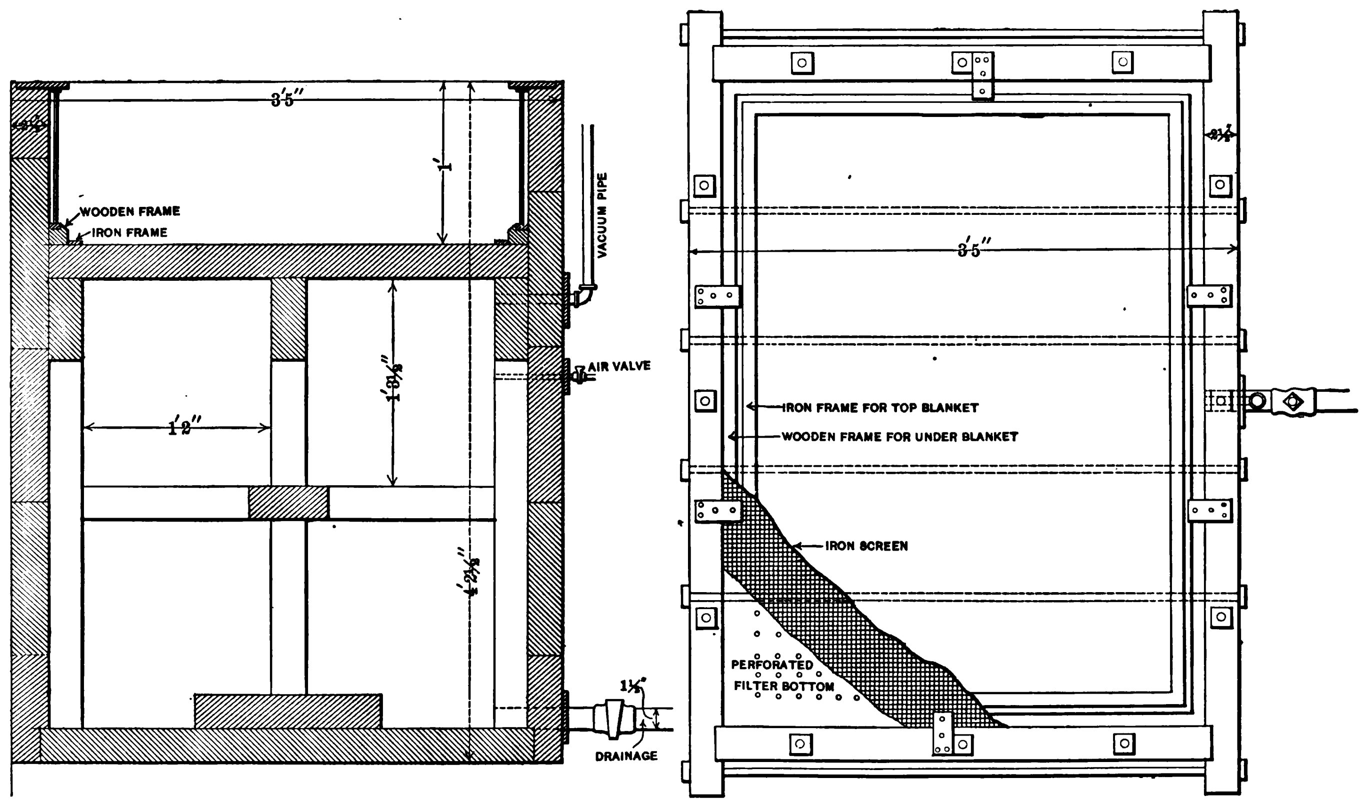


FIG. 11.—FILTER BOX.

seems to resist very well the action of dilute acid) or of any other good tank material. In some localities lead-lined tanks are used. It is essential that a wooden tank should frequently be painted with some good preservative—such as a paraffine paint. A redwood tank, with staves and bottom 3 in. thick, has been in use for two years at the Standard Works, and shows only slight traces of being attacked by acid. From present indications it will last several years longer.

There appears to be considerable advantage in having this tank broad and shallow. A more intimate contact between the slimes and the acid is brought about by this greater extent of surface, and thereby a more rapid dissolution of the zinc element in the slimes; the annoying tendency of the mass to boil over (a noticeable feature of zinc reduction in narrow tanks or boxes) is corrected; and manipulation of the slimes, cleaning out the tank, etc., greatly facilitated.

Filter-Box.—The filter-box (Fig. 11) is a modification of the apparatus used in agitation plants in New Zealand and elsewhere.

It consists of a carefully made water-tight box, 4 ft. high, 3 ft. long, and 2 ft. wide, constructed of 2½-in. pine planks. The interior of the box is divided into two compartments by a filter-bottom, which lies 12 in. from the top. This filter-bottom, which is 1¾ in. thick, and perforated to within 1½ in. of its edge with ⅜-in. holes, 1½ in. apart, is supported by a center post and four corner posts; and any possible collapse of the sides of the box is forestalled by braces running at right angles to the center post. To resist the strain imposed by the vacuum pump, the whole box is further strengthened by a system of ½-in. bolts, running through the timbers from top to bottom, and across the ends.

Details of False Bottom.—The perforated area of the false or filter-bottom is covered by a piece of heavy wire cloth, about 15 meshes to the inch; the object being to prevent the superimposed blanket filter from sagging down into the perforations, and to increase the filtering area by preventing close impact of the blanket with the surfaces between the holes. On top of the screen is placed one or more thicknesses of heavy mill blanketing, as the case requires. The lower blanket is pressed into place by means of a wooden frame with sides 1½ in. wide. The blanket is cut large enough to admit of its being pulled up around the frame. Into the top of the frame, in the middle of each side, is sunk a strip of iron ⅛ in. thick and 12 in. long. In the center of each

strip is a square hole, to receive the squared end of an upright rod. The upper end of this rod is threaded, and passes through a strip of iron which projects over the inner edge of the top of the box. The rod is provided with a nut, which, when tightened up against this top strip of iron, compresses the wooden frame tightly over the blanket. The frame, with one of these rods on each side, prevents fine slimes from passing down into the compartment below the filter bottom.

Filter Blankets.—Ordinarily one blanket will suffice; but it will be found more convenient to use a second, which may be easily removed and washed, while the first may remain stationary through several clean-ups. The second blanket may be stretched in place by means of an iron frame, with sides 1 in. wide, which should be made to fit accurately within the wooden frame which holds the under blanket.

Valves.—At the bottom of one side of the box is a 2-in. Globe valve, for the discharge of clear fluids accumulating under the filter; and just beneath the filter is the entrance of the ½-in. exhaust pipe coming from the vacuum pump. If steam is used for vacuum purposes, a 1½-in. steam ejector may be connected. Near the entrance of the vacuum pipe is an escape valve, which may be opened to release the pressure, when, for any reason, such as changing the blankets, etc., filtering is temporarily interrupted. For convenience, a gauge may be connected to the side of the box, to indicate the height of liquid under the filter. This liquid should be drawn off before the surface reaches the height of the vacuum pipe.

Settling Tank.—The settling tank should be large enough to hold seven or eight times the contents of the acid tank. Its proportions will depend a good deal upon the space allowed for the clean-up room. At the Standard Works it stands just outside of the main building, for lack of room inside. The advantage is obvious, however, of having such a tank under cover, and of placing it as near as possible to the acid tank. Convenient dimensions would be 7 ft. deep and 8 or 9 ft. in diameter.

Drying Precipitates.—The refined gold precipitate may be dried on an open brick furnace, in an ordinary muffle, or (if there is any considerable amount of quicksilver present) in a retort furnace.

The size of the open furnace will depend upon the size of drying-pan required. A pan for holding 2,000 oz. of precipitates (dry weight) should be 3 ft. long, 2 ft. wide, and about 8 in. deep, and

constructed of $\frac{1}{8}$-in. iron sheeting with riveted joints. The furnace may be provided with a heavy iron grating of suitable dimensions to support this pan. If such a method of drying is resorted to there should be a Russia-iron hood suspended over the furnace, which can be let down so as to inclose completely the drying-pan. The top of the hood may be connected with a pipe, for carrying off irritating or poisonous fumes, such as the vapors of mercury and sulphur. (The odor of sulphur should scarcely be noticeable if the precipitates are thoroughly washed after acid treatment.)

Retorting Precipitates.—Where any quantity of mercury is precipitated from the solution in the zinc-box, it may pay well to retort the precipitates, a procedure adopted at the Marion Cyanide Mill in Utah, and recently at the Eureka Cyanide Works, Nevada, where a considerable amount of quicksilver is recovered at each clean-up.

Hot Water Boiler.—Adjacent to and connected with the furnace is a water boiler for providing hot water for the acid tank and filter box.

Assaying and Melting Rooms.—If a plant is isolated, that is, not directly connected with other metallurgical works, assaying and melting rooms must be annexed. These two rooms should be distinct, and separated at some distance from each other, owing to the great danger of assay samples being "salted" by the finely divided zinc precipitate.

CHAPTER VI.

ARRANGEMENT OF PIPES, VALVES, ETC.

It may be contended that certain details in the construction of a cyanide mill, such as the arrangement of solution pipes, etc., might be more fitly left to the ingenuity of those in charge of construction. Yet it will be admitted, I think, by all who have had much experience with the process, that there is no greater source of annoyance and inconvenience in a mill than a clumsy arrangement of the means of conveying solutions, or one more difficult to remedy. The importance of a correct system at the start is obvious when we reflect that the chief concern of a mill-foreman is in testing solutions and directing them into their proper channels by manipulation of the different valves. Therefore the simpler and more convenient the system the more expeditiously can he perform his work.

It is a common experience in cyanide mills to find that a pipe and valve system, which appeared at first very convenient and ingenious, became almost completely altered in time, as new necessities arose and better facilities suggested themselves. Every mill-foreman is probably familiar with the drudgery of tearing up and altering an old system of pipes and valves.

Several years' experience in Bodie, where the process has attained a high degree of practical efficiency, has resulted in a very good system of conveying cyanide solutions and water in a mill, which the writer feels justified in describing briefly.

Water Supply and Pipes.—First, as to the water supply. The force of water should be great enough to insure a powerful discharge from the nozzles of the sluicing hoses. For this purpose a tank or reservoir may be arranged, 30 or 40 ft. above the mill, with a discharge reducing gradually from a 6-in. to a 3-in. pipe. In the mill it will be found most convenient to extend the 3-in. pipe over the center of the vats, with two hose attachments for each vat—

one at each side just within the staves. The hose may be directly attached to 2½-in. iron plug service-cocks. Heavy 5-ply rubber hose (2½-in.) at least 20 ft. long, with nozzle having a ¾-in. discharge, is suitable for sluicing out a 75-ton vat. For smaller vats a 2-in. hose with ⅝-in. discharge might be used.

After sluicing, the hose is disconnected and the same service-cocks used for regulating the supply of wash-water to a vat. Sluicing out will require approximately, with a head of 30 or 40 ft., about 100 gal. of water to the ton of tailings. This will, of course, vary with the character of the material. Fine or slimy tailings and very coarse stuff require considerably more water than material of medium fineness. In addition, a 75-ton plant will require 4,000 or 5,000 gal. more per day for wash-water used in displacing solutions.

Where the material is of variable fineness, a tank for providing sluicing and wash-water to a 75-ton plant should have a maximum capacity of about 12,000 gal. In some instances direct pumping into the sluicing hose has been resorted to where it has been found impossible or inexpedient to build a reservoir.

Pipe to Storage Tank.—A branch from the main water pipe should be run to the top of the strong-solution storage tank for supplying water needed in making up the original solution.

Solution Pipes and Valves.—For vats 20 ft. in diameter, 2-in. solution pipes may be used; for vats 25 ft. in diameter, 2½-in.; and for vats 30 ft., 3-in. It is important that the feed pipes should be large enough to insure a rapid discharge of solution into the body of the ore.

The two feed pipes from the strong and weak-solution storage tanks may run side by side above the vats, near enough to the edge to be conveniently reached by the operator.

Drop Pipe for Strong Solution.—As considerable time is saved by saturating a vat-charge from the bottom, the main strong-solution pipe may be provided with a drop pipe (Fig. 12) extending down to just beneath the filter cloths. A wooden flange should be placed around this pipe at the point where it pierces the filters, to prevent the sand from getting under the false bottom. This drop pipe (which passes down close to the staves) is provided with two valves, by means of which solution may be introduced at the bottom or at the top of a charge, as required.

The weak-solution pipe is simply provided with a valve for running solution on top of the charge.

PLATE VI.

INTERIOR OF VICTOR PLANT, BODIE, CAL.

Showing leaching vats, bridge over vats, and main strong-solution pipe.

The operator manipulates these valves from a platform built around one side of the vats (the side nearest the storage tanks) and raised at a suitable height from the ground. This platform is connected by stairs with the office and storage-tank shelf; and with the leaching-vat shelf by means of another stairway, erected between the second and third vats (see Plate I.).

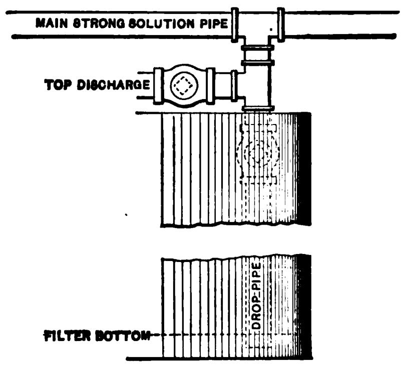


FIG. 12.—DROP DISCHARGE PIPE FOR STRONG SOLUTION.

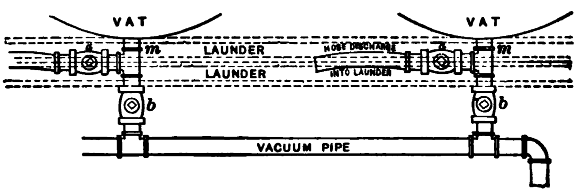


FIG. 13.—PLAN OF VACUUM AND DRAINAGE PIPES.

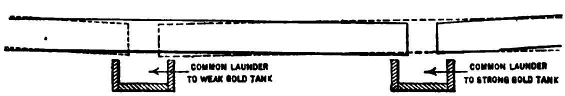


FIG. 14.—ELEVATION OF LAUNDER SYSTEM FOR CONVEYING SOLUTION FROM VATS.

Discharge or Outlet Pipes.—The arrangement of discharge pipes may be best understood by referring to Fig. 13. It is so devised that the vacuum pump may operate through the same pipe that discharges gold-bearing solutions into the launders, the alternations being effected by the valves *a* and *b*. The section of pipe *m* is provided with a deep thread 3 in. long (six threads to the inch),

which screws into the vat, protruding inside just beneath the filter cloths. A more secure method of fixing this discharge pipe to the vat would be by means of a flange, but this is hardly feasible owing to the position of the lower hoops. At the point of discharging into the launders a short piece of hose is attached, which may be shifted from one launder to the other, according to the destination of the solution.

Launders.—The two launders (for strong and weak gold-bearing solution), which should be constructed of clear, seasoned pine boards, are 4 in. wide and 4 in. deep. The edges of the bottom piece should be accurately dressed and covered with a heavy coat of paraffine paint before being attached to the sides. The whole may then be put together by means of screws 4 in. apart, and both inside and outside thoroughly painted. Each launder is divided into two segments (Fig. 14) inclined toward a common discharge center. Under each discharge is a cross launder 8 in. wide, which carries the solution running from either side to its proper gold tank. These launders should be carefully constructed, or they will cause no end of annoyance by leaking.

Another system is in use for leading solutions from the vats, consisting of a separate, continuous pipe from each vat to the gold tanks. This necessitates attaching the valves at the ends of the pipes over the gold tanks, instead of at the points where solution issues from the vats. In a small plant such a system is open to several objections: (1) It requires the use of valves in shifting solution from one gold tank to the other; (2) it complicates the vacuum pump attachments by requiring a separate pipe and valve for each vat; (3) it requires several hundred feet more of pipe, without affording any advantage over the launder system. In a large plant, however, where the vats cover a vast area of ground (as in certain South African works) a separate discharge-pipe from each vat is required. All the vats then discharge at one point, and samples of solution can be conveniently taken.

Launders are objected to as being a troublesome source of leakage, which they unquestionably are, if poorly constructed. It is also claimed that they take up too much room. If so, then space should be allowed for them in constructing a small plant, just as calculations are made for the space occupied by tanks and zinc-boxes.

Valves.—Ordinary iron plug service-cocks may be used throughout a plant. They should be so placed, wherever possible, that solu-

tion leaking from the plug may drain into a tank or launder. On account of the active affinity of cyanide solution for copper, it is not advisable to use brass valves or valves containing any brass.

Pipe.—Lap-welded iron gas pipe is generally used. It is important that the material should be the best, so that the joints may be made absolutely tight wherever the pipe is fitted. Paraffine paint may be used on the threads as a precaution against leakage. The pipe should be dipped in asphaltum, to prevent contact of cyanide solution with the iron.

Discharge from Gold Tanks.—Each zinc-box should be provided with a separate discharge from the gold tanks, an arrangement to be preferred to a single branching discharge. The discharge nipples and valves may best be connected with the tanks by means of bolted flanges. If the nipples be introduced directly into the staves they are apt to work loose from long-continued manipulation of the valves; an objection which does not hold good for the leaching tanks, where the discharge pipes are firmly held in place by the transverse vacuum pipe. Iron globe valves are perhaps better adapted than the plug valves to feeding the zinc-boxes; they can be more easily regulated to produce slight variations in the stream. It is advisable to have an extra valve attached to the bottom of each gold tank, to which a hose may be connected for draining off into the sump, or elsewhere, the various precipitates thrown down during leaching.

The zinc-boxes may be discharged through a 2-in. nipple introduced at any point in the last false compartment.

Centrifugal Pump.—The centrifugal pump (which is perhaps the best pump adapted to low head and large capacity), for returning the solution to the storage tanks, may be placed between the two sumps with a T connection on the suction, by means of which solution may be drawn from either tank. The bottom of each suction pipe should be provided with a foot-valve. The discharge pipe may be carried directly along the roof of the zinc-room, and over the leaching vats to a point between the two storage tanks, where another T connection should be fitted and provided with suitable valves for discharging into either tank. It is advisable to use long-radius elbows in the suction and discharge pipes, to prevent, as far as possible, losses from friction in the momentum of the stream.

Vacuum Pump.—There is considerable choice in the form of vacuum producer, depending in each locality upon which is the

cheapest. Where fuel is expensive the steam ejector for hastening percolation would obviously not be suitable. Where there is an abundance of available water, some form of hydraulic ejector, such as is used at the Eureka Cyanide Works, in Nevada, might be introduced. Ordinarily, however, a suction pump is the only choice. Excellent results are obtained in Bodie from a Gould vacuum pump working directly on solutions without a receiver. The receiver is a boiler-shaped chamber, interposed between the exhauster and the leaching vats, and usually placed above the gold tanks. It accumulates the solution which would otherwise pass into the pump. It is usual to have two receivers, one for strong, and one for weak solution. The receiver has the disadvantage of requiring to be discharged periodically as it fills with solution, whereas a suction pump, operating directly on the leaching vats, requires only a minimum of attention.

The suction from the vacuum pump may be connected with a receiver or directly with one end of the vacuum pipe running in front of the leaching vats (Fig. 13). Its discharge into the gold tanks is fitted with a T connection, similar to the ones described above for shifting the discharge into either weak or strong gold tank. A branch vacuum pipe ($\frac{1}{2}$ in.) may be run from the main vacuum suction pipe down to the filter box in the clean-up room, with a valve conveniently placed.

In Bodie practice the steam ejector is used to exhaust air from the filter box, but only because a steam boiler, used for other purposes, happens to be near at hand. Where steam is dispensed with in favor of some cheaper form of power the vacuum pump may be applied as I have explained.

At the more recently constructed of the two Standard plants, a gasoline engine operates, among other things, a Johnson filter press, which partially dries the slimes preparatory to acid treatment; these slimes are then taken to a common clean-up room at the other plant, where the process is completed. The same work might be accomplished by the Gould vacuum pump operating on a filter-box if it were not that the Johnson press (having been found unavailable for drying slimes after acid treatment) was relegated to this use, rather than laid aside.

Pipe Connections in Clean-up Room.—The settling tank should be fitted with a series of valves or plugs along the side for drawing off clear liquid from different levels. The bottom of this tank

may be slightly inclined, and fitted with a service-cock for discharging the accumulated mass of precipitate.

It is desirable to have a hot and cold water pipe (1 in.) connection with the acid tank, and a 2-in. nipple and service-cock attached to the bottom of the same tank for discharging the refined precipitate into the filter-box.

CHAPTER VII.

PREPARING ORES OR TAILINGS AND CHARGING VATS.

Ores.—The preliminary treatment of ores adapted to cyaniding consists in (1) wet crushing by stamps, and the saving of the coarse gold by plate or pan-amalgamation, and by concentration; (2) drying the ore, and crushing in ordinary rock breakers or patent rolls.

The cyaniding of wet crushed ores, after intermediate or direct filling of vats, is complicated by mechanical difficulties, which are discussed in Chap. VIII. The most conspicuous success of the process, however, has been in the treatment of tailings in South Africa, where many of these mechanical difficulties are being rapidly overcome.

Where amalgamation can be dispensed with, rolls and rock breakers are found to answer the purpose of crushing much better than stamps. Dry crushed ores are more easily handled, and can be more evenly distributed in vats than the wet crushed material. Immense plants for the preliminary drying, crushing, and classification (and in some instances, roasting) of ores have recently been constructed in Colorado and Utah, in connection with cyaniding works. This preliminary treatment necessarily varies greatly in different mills, according to the character of the ores; in another chapter will be found descriptions of some of the more elaborate of these works in Utah, Colorado, and elsewhere.

In the simpler cases, the ore is broken by one of the various forms of crushers (the Dodge and Davis crushers being commonly used) and then passed to rolls for finer crushing. The Krom, Davis, and the Wall's corrugated rolls are used. From the rolls the ore is classified by a system of screens—the oversize being returned for further reduction. The material thus prepared is then gathered in bins, and conveyed to tanks either by cars or belt conveyors.

In other localities the system is more complicated. The ore,

after being subjected to coarse crushing, is conveyed to driers, either kilns or rotary iron tubes. Thence it is conveyed to another system of crushers, rolls, and screens, for further reduction and classification. It is then taken either to roasting furnaces or to the ore bins. In Colorado practice a classification is made between the oxidized tulluride ores and the unoxidized; the latter pass to the roasting furnaces, the former to the storage bins. Furnaces which seem to be coming into high favor in Colorado are the Brown Horseshoe, and the Pearce Turret furnaces, based upon the principle of a wide annular hearth, and rabble arms moving around a central pivot.*

At the Mercur Mill, Utah, the case is a simple one; the ore merely requires coarse crushing preliminary to cyanide treatment.

Tailings.—In mill reservoirs the tailings are usually found coarsest at the site of discharge from the mill, and finer as we approach the extreme end of the pond, where the slimes are deposited. At the present time one method of solving the difficult question of slimes treatment is by mixing the coarser and finer elements. The tailings may either be mixed *in situ*, or on the grating through which they are dumped into the vats. By dumping alternate loads of coarse tailings and slimes from a tramway or through a grating, the loads become pretty well intermixed in a vat, but not the particles of coarse and fine; and by this method we neither get a uniform percolation nor the best extraction. If mixed in the reservoir, the tailings should be thoroughly plowed and harrowed, and all lumps of slimes broken up by some form of crusher or disk-harrow. When well dried and pulverized they may be scraped up into heaps; the best method is to have two scrapers at work, one gathering in the fine tailings and one the coarse. In shoveling from such heaps into wagons the material becomes pretty well mixed, and the process is practically completed when the load is discharged. Another method suggests itself—taking half a load from one part of a bed, and the remainder from another. This method, however, is awkward, and would require additional teams of horses; the prevailing system being to hitch the horses to a loaded wagon, and to leave behind the empty one, to be filled during the interval of hauling. Much time would be lost in moving the wagon about to different parts of the bed for a single load. With the system of hauling in cars, or by some form of endless conveyor, this last-mentioned method might be even more clumsy.

* For description of Brown furnace see *Engineering and Mining Journal*, July 4, 1896.

The best practice would, therefore, seem to be plowing, scraping, and as thoroughly as possible mixing the tailings before loading.

This, however, is impracticable where the coarse and fine tailings are derived from different sources, perhaps at some distance apart. At the Eureka Cyanide Works, Nevada, all tailings are discharged into a common hopper, where they are thoroughly mixed by a revolving screw, and thence conveyed to the vats by means of an endless belt-conveyor (see illustration).

Tailings Containing Organic Matter.—Old grass-covered reservoirs or accumulations lying along the banks of creeks or rivers frequently contain a considerable admixture of organic matter, such as roots, fragments of decaying brushwood, etc., which have to be pretty thoroughly removed before profitable cyaniding is possible. Whether the necessary preliminary treatment will pay or not must be determined for each case. If the material lies in scattered, shallow accumulations, it may be shoveled up into heaps, through a suitable screen, just as coal is screened; if in beds compact and deep, it may be shoveled directly into screen-covered wagons. The size of screen will obviously depend upon the character of the material. In the neighborhood of Bodie it has been found profitable to gather up tailings from old grass and willow-covered deposits along the creek banks. For separating the roots, etc., a screen of ¼-in. mesh was found suitable. In this particular case, however, the employment of cheap Indian labor is all that makes the system profitable.

Alkaline Wash.—In some instances, tailings containing a large amount of free acid are treated to an alkaline wash before solution is applied. Such a system, however, involves considerable time, and should be resorted to only in special cases, where exceptional quantities of alkali are required, or where the material treated contains deleterious salts which might foul, or otherwise interfere with the efficiency of the solution, if not removed by a preliminary wash. When found necessary, a separate tank, with a capacity equal to about one-third of that of the leaching vat, may be provided for this alkaline wash, and a sump for gathering it as it leaves the vat.

Mixing with Lime.—In most instances the most economic method is to mix powdered caustic lime directly with the tailings before, or at the time of, delivering them into a vat. This may be done either by scattering the lime over the wagon or car before dumping, or throwing it through the grating after dumping. A

sufficient quantity for a day's run may be slaked; a preliminary which prevents dusting and does away with the necessity of crushing the lime to a uniform fineness.

Sampling.—The discrepancies often observed between the yield indicated by assays and the actual bullion returns have been largely attributed to imperfect sampling of the material before and after treatment. If the tailings are poorly mixed, and contain lumps of slimes of higher grade than the coarse tailings, it is quite impossible to obtain a reliable sample with the ordinary sampling rod, or tryer, used for the purpose. The result is apt to indicate too low a value. On the other hand, the discharge sample taken in the same way is very apt to indicate a result too high, on account of the partly disintegrated lumps of slimes adhering to the tryer, and constituting too large a proportion of the core. From tailings properly mixed, however, there should be no very serious errors from sampling.

The method practiced in Bodie is simple, and has given good results. From each wagon-load several cores are taken with a Russia-iron tryer 18 in. long and 1½ in. wide. The gross sample thus accumulated (about 200 lb. for a 75-ton charge) is thoroughly rolled, mixed, and quartered down, and from this lot a final sample of 200 gm. is taken. This sample is dried, reweighed, and the percentage of moisture determined. It is then subjected to a sizing test through an 80-mesh screen and assayed.

A sample of the residues after cyaniding is taken with a 5-ft. tryer, at first introduced vertically 20 or 30 times through the charge; and later on, when a trench has been sluiced through the center of the charge, horizontally. This sample is dried, quartered down, and assayed.

In spite of the most careful sampling and assaying, however, discrepancies will occur, which must be laid to the impossibility of getting a perfectly correct sample from so large a body of material, and from the extreme difficulty of obtaining a correct estimate of the original moisture. Accurate weighing of the charge is not always practicable, and in such cases should be allowed for in accounting for a variation between assay indications and the actual yield.

Weighing.—The wagon-loads before entering the plant are weighed on platform scales, and by this means the exact net weight of each charge is recorded, the moisture being determined from the assay sample.

Charging a Vat.—The discharge of material into a vat should not be concentrated at any one point, but as evenly as possible distributed, so that the whole surface of the charge rises uniformly. This may be accomplished by means of the grating already described, which permits of the wagons being discharged at any point desired.

Depth of Charge.—The proper depth for a charge has been variously placed at from 6 in. to 12 in. below the top of the vat. As a rule, a vat may be safely filled to a level with the top; for when saturated with solution it will settle down several inches. However, this settling will vary with the material; and the depth to which a vat may be filled will have to be determined in each case. After subsidence there should be left a space of about 10 in. above the charge for accommodating solution and wash-water.

Quantity of Stock Solution Required.—The quantity of strong solution (in tons) in circulation should be about equal to the daily ore capacity of the plant. In mixing the first stock solution at the commencement of operations, the strong-solution storage tank and the strong-solution sump may be filled at once; and when this supply is exhausted the storage tank may be half-filled again.

The proper strength for strong solution having been previously determined by laboratory tests, the next step is to calculate the liquid capacity of storage tank and sump, in order to get at the total quantity of cyanide to be used.

To Calculate Capacity of Tank.—The liquid capacity of a cylindrical tank in gallons or tons may be calculated as follows: Capacity of tank in gallons equals cubical contents $\times 7.48$. There are 240 gal. in a ton of water; hence the capacity in gallons, divided by 240, equals tonnage. For example, the cubical dimensions of a tank 12 ft. in diameter and 10 ft. deep are, in round numbers, 1,131 cu. ft. Its capacity in gallons would be $1{,}131 \times 7.48$, or 8,460 gal. Its tonnage would be $8{,}460 \div 240$, or 36 tons. A solution containing 0.2% cyanide would require 4 lb. cyanide to 1 ton (2,000 lb.) of water; hence it would require 144 lb. of pure cyanide of potassium to stock a tank 12 by 10 ft. with strong solution at the above strength. The best cyanide manufactured is 98 to 99% pure, so that in using it we may, for all practical purposes, assume it to be a chemically pure article.

Similarly, a solution to contain 0.3% cyanide would require 6 lb. to the ton, and so on.

CHAPTER VIII.

THE LEACHING PROCESS.

First Stage: Saturation.—The surface of the ore or tailings having been leveled off and the cyanide solution of proper strength prepared, the next step is to saturate the charge. The solution may be run through a full pipe from the storage tank into the bottom of the vat by means of the drop pipe, shown in Fig 12. Under ordinary conditions it will require about an hour to saturate a 75-ton vat. The solution should be run in until the whole surface is covered to a depth of 2 or 3 in.

Second Stage: Preliminary Soaking.—The charge should then be allowed to soak for about two hours, so that the solution may penetrate to all points, and establish a more uniform density throughout the charge, preparatory to leaching.

Third Stage: Preliminary Leaching.—As soon as the valve at the bottom of the leaching vat is opened and percolation, or leaching, commences, the outgoing solution should be tested for strength of cyanide. If the charge contain much moisture, the first solution running out will be weak; if less than one-half the strength of the standard solution it should be run into the launder conducting to the weak gold tank. The variations in strength of the first outgoing solution are apt to be capricious. The solution should be frequently tested until it begins to show a steady rise in strength.

Rule for Differentiation of Strong and Weak Solutions.—All cyanide liquors of more than one-half the strength of the standard strong solution should be run into the strong gold tank; all of less, into the weak tank. This rule may be varied in exceptional cases when, for any reason, the strong or weak solution accumulates faster than it can be disposed of. In such a case the differentiating point of strength may be raised or lowered until the balance is again established.

For instance, if the standard strength be 0.2%, then all solu-

tions containing from 0.1 to 0.2% cyanide are run into the launder leading to the strong gold tank; the average strong sump solutions will therefore be from 0.13 to 0.16% cyanide. Solutions under 0.1% in strength are run to the weak gold tank, whose solution will average from 0.05 to 0.07% cyanide.

Upon the strict observance of this rule and the frequent titration of these variable solutions, much of the practical success of a plant will depend.

The preliminary percolation may be continued until practically all the original moisture in the tailings is displaced; in other words, until the outgoing solution reaches within 0.02 or 0.03% of standard strength. To wait until a solution of standard strength appears would be unnecessary; such a point would not be reached until the action of all the cyanicides in the tailings had ceased.

Until all the moisture is displaced the tailings do not really get the benefit of a solution of approximate standard strength; it is not until the fourth stage in the leaching process commences that the mass of solution in a vat is of the best available strength for extraction.

Fourth Stage: Actual Soaking.—If the charge be of pretty uniform density, actual percolation may be commenced at once; if not, a second soaking of several hours will be of decided benefit. It allows the cyanide-consuming materials in the tailings to complete their action, so that when percolation follows the strength of cyanide continues uniform until weak solution is applied. However, as the necessity of oxygen in hastening the dissolving action of cyanogen has been demonstrated, a limit will be reached beyond which a soaking solution will become less efficient than a percolating solution. This limit should be determined in the laboratory. Ordinarily it is not well to prolong the actual soaking beyond six hours. In many instances, soaking or maceration is omitted altogether, and percolation with standard solution follows at once upon charging the vat.

Fifth Stage: Actual Percolation.—The time required for percolation with solution at standard strength must be determined by testing. The freer the gold, and the finer its state of division, the less the time required. It will be found to vary greatly with different conditions. At the Standard Works in Bodie, the best economic results have been obtained by soaking the tailings 20 hours with strong solution (after moisture is displaced) and then adding weak solution. At the Mercur Mill the best results were

obtained by using a series of strong-solution washes, each time draining off the solution before adding fresh. The time of percolation will be found to vary between 12 hours and several days. A good working method is as follows: The solution being partially drained from the charge, after soaking, a fresh quantity of solution is run on top of the tailings to a depth of 8 or 10 in. This is allowed to disappear beneath the surface. In an hour another quantity is added.

This succession of solution charges is continued until the prescribed limit of time for percolation has been reached. By thus partially draining the vat at intervals, a free access of air to the leaching material is insured, and the dissolution of gold accelerated.

Draining.—In some mills a vat is thoroughly drained before a succeeding charge of solution is added. This precaution would seem to be superfluous. A great amount of time is consumed in completely draining a vat, which might be better employed in bringing fresh solution into contact with the tailings or ore. If drainage is carried too far the material exhibits a tendency to pack, retarding further percolation. It will be found in most cases that a sufficient access of oxygen may be brought about by allowing the solution to disappear for an hour or so below the surface of the charge before applying fresh liquid.

Sixth Stage: Displacement of Strong Solution by Weak.—At the commencement of operations in a new plant water may be used to displace the strong solution until sufficient weak solution has accumulated. If water is used, it may be kept flowing on top of the tailings until silver nitrate indicates the presence of little or no cyanide in outflowing solutions. If the test shows only 0.03 or 0.04% cyanide, it is safe to assume, in most instances, that the solution carries only an insignificant value. Whether it will pay to continue the operation until all the cyanide is displaced is a question to be determined by practical experiment. It is obvious that if a 0.03% solution contain only only 30 or 40c. in gold per ton, it would not pay to continue the displacing process unless the total amount thus saved would outweigh the additional cost of treatment.

When sufficient weak solution has accumulated in a new plant, it should be used to displace the strong. The average strength of solution in the weak sump will be found to be about one-third that of the standard strong solution. If, then, a solution of, say, 0.07% in strength is run on a charge, it is used continuously until the outgoing solution falls to 0.07%.

Seventh Stage: Displacement of Weak Solution with Water.—Water may then follow, and the displacement by water be continued as far as is found expedient.

It is always advisable to use as little water as possible, to prevent a needless accumulation of weak solution.

In some mills it is customary to accumulate no weak solution whatever beyond what is required for working purposes, the surplus in each case being got rid of with the spent tailings. This system is usually resorted to where, owing to difficulty in precipitating from weak solution, the latter is used as a wash. It is popularly thought to be a great merit in the operation of a mill that no solution is run to waste! The mere fact that something is wasted is enough to suggest at once, to the superficial observer, some glaring fundamental defect. As a matter of fact, the moisture present in most leaching material, and the dilutions caused by wash-water, make an unavoidable increase in the total quantity of circulating solutions, a surplus which must be disposed of either by running it to waste, after passing it through zinc-boxes, or discharging a given amount of it each day with the spent tailings. The latter method, though time-saving, is open to the objection that it causes the loss of a considerable amount of gold-bearing solution. If the bulk of this solution is displaced by water, and run through "waste zinc-boxes," practically nothing is lost but the cyanide it contains. Such solution may be disposed of, whenever there is an unmanageable surplus, by running it to waste through a service-cock, provided for the purpose, at the bottom of the weak sump.

This system, while certainly defective, is at the present time the best we have. A perfect system would be one in which all solution could be displaced from a vat before discharging, and none whatever run to waste. What is really needed is some cheap means of recovering the cyanide from this surplus of weak solution. The method suggested by Professor Christy,* of precipitating the potassium cyanide by means of zinc sulphate, might be applied in practice: whether the cyanide thus recovered would pay for the labor involved and the chemicals consumed is yet to be demonstrated.

As already pointed out, this part of the leaching process (displacing strong solution with weak solution and water) is apt to be prolonged and tedious. One reason for this is that the tailings be-

* "Transactions of American Institute of Mining Engineers." vol. xxvi., p. 766.

come more compact toward the end of the operation and the rate of percolation slower.

Another is that the displacement does not appear to take place uniformly. If there were a hard and fast line of division between the displaced and displacing solutions, they would each be equal in quantity. But experience in cyanide mills shows that it sometimes requires half as much again, or even twice as much weak solution, to displace the strong, as is theoretically indicated. This is perhaps partly to be accounted for on the supposition that instead of a simple mechanical displacement, there occurs in reality a mixture of the strong and weak solutions, due to the density of the mass of tailings not being uniform, and to the existence of channels throughout. It is also possible that the retention of strong solution may be greater at some points than at others, as in lumps or strata of slimes, and its displacement, in consequence, slower.

The prevailing system of displacing solutions in cyanide works in South Africa and New Zealand appears to be somewhat different from the one just described. The weak solution and water-washes are introduced in succession, as first, second, third, and so on; each containing a prescribed quantity of liquid, according to the capacity of a plant. A wash is run into the vat and allowed to drain, and another follows.

If the conditions were absolutely the same in every vat-charge of ore or tailings, this system might be universally depended upon, after determining from a series of test-vats, just the amount of displacing solution required. But inasmuch as in the vast majority of instances certain conditions will be found to vary with almost every vat, it would certainly appear as if a more reliable method would be to continue displacing independently of the number of washes, until silver nitrate showed the solution to be sufficiently poor in cyanide.

This system of frequent testing may be objected to on the ground of its involving much tedious, troublesome work. In a plant containing a large number of small vats it would probably be a nuisance; but we are assuming that the greatest efficiency lies in a small number of large vats.

An absolute uniformity of conditions in all the vats in a plant is absolutely impossible to attain. Charges of tailings loaded from about the same place in a reservoir, and apparently of the same general character, will be found to exhibit, during the leaching process, certain inexplicable differences in rate of percolation, time

required for displacing solutions, and so on, which will puzzle the operator and require his undivided attention and study. The operations in each vat should be observed as closely as if it were a mill by itself; for only by close attention to all such matters of detail can the best results be obtained. While the operator should so aim to systematize the operations in a plant as to attain a maximum of efficiency with the least expenditure of time and labor, there are some things which cannot be neglected or only half done without seriously affecting the results. No unalterable rule can be relied upon for the exact amount of weak solution and water required to displace the strong solution in a vat; frequent testing of outgoing solutions is our sole reliance, and ought to be carefully performed.

Stannous Chloride Test.—In some plants this test is used to determine when the final solutions are sufficiently weak to justify discharging the residues. It depends upon the fact that stannous chloride ($SnCl_4$) in the presence of simple solutions of gold gives a characteristic violet reaction known as the "purple of Cassius." The test is extremely sensitive, revealing the presence of gold "in a solution containing one part of gold in 500,000 parts of water, while by special means the presence of one part of gold in 100,000,000 parts of water can be detected" (Rose). If applicable to auro-cyanide solutions it would be an excellent means of determining quantitatively the amount of gold in solution at the various stages of percolation. Tubes containing solutions of gold in distilled water, to which stannous chloride has been added, representing certain known gold values per ton, might be used as guides; and by comparison with these the value of any cyanide liquid could be at once estimated. But unfortunately, owing to the great complexity and instability of the cyanide solutions used in actual practice, this test does not give results sufficiently uniform to be reliable.

The silver nitrate test is accurate enough for all practical purposes in determining when a vat should be discharged.

If the strong solution in a plant contain 0.2% cyanide, averaging in gold value about $4 per ton, it follows that if the same solutions be diluted with the displacing liquids so that it contains only 0.04% cyanide, its gold value per ton has been correspondingly reduced. The limit of this final displacement will, as already explained, depend wholly on the cost of treatment, and can only be determined for each case by a series of tests. At the Standard Works in Bodie,

where the strong solution averages about \$4 per ton in gold, the displacement is carried on until the outgoing liquid tests 0.04% cyanide. A succession of exhaustive experiments showed such a diluted solution to be worth from 25 to 35c. per ton in gold. When a 75-ton vat is finally drained, preparatory to discharging, the contents presumably contain about 15 tons of solution, of a total value of from \$3.75 to \$5 in gold. This retained moisture is discharged with the residues, because it would not pay to displace it.

This point has been insisted upon at some length because, obviously, it is a matter of no little importance to determine just when the economical limit of time has been reached in the treatment of a charge of ore or tailings. There is perhaps no point in the whole cycle of cyanide operations which requires so much accurate calculation, or one upon which more important results depend.

Discharging a Vat.—Before sluicing out, the wash-water should be drained off sufficiently to permit of vertical sampling with a tryer. The charge may then be covered with water, and allowed to stand until the mass is thoroughly saturated, when the central plug, or plugs, may be drawn. The rush of superincumbent water down through the discharge holes carries a considerable bulk of the residues with it, and so expedites sluicing. If tailings have to be shoveled out, this procedure, for obvious reasons, is omitted, and the final wash-water drained off until the tailings are dry enough for convenient shoveling.

Resumé of Leaching Operations.—(1) Saturating with strong solution at standard strength; solution to be introduced into the bottom of vat through drop-pipe.

(2) Preliminary soaking with strong solution one or two hours, to establish a more uniform density, and to allow solution to penetrate to all points.

(3) Preliminary percolation, or displacement of solution in vat (diluted by moisture in tailings and reduced in strength by cyanicides) by standard solution; the percolation to be continued until the outgoing liquid tests within two or three points of standard strength.

(4) Actual soaking for about six hours with the best available solution, *i. e.*, at standard strength.

(5) Actual percolation (duration variable, and to be determined by laboratory investigations) with solution at standard strength.

(6) Displacement with weak solution until strength of outgoing liquid is equal to strength of weak solution in storage tank.

(7) Displacement of weak solution with water until outgoing liquid shows 0.03 or 0.04% KCy (the exact point to be determined in laboratory).

(8) Discharging by sluicing or shoveling.

Unwieldy Accumulation of Strong Solution.—If, according to this system of leaching, solutions between 0.10 and 0.20% KCy are run into the strong gold tank, and those between 0.03 and 0.10% into the weak, it is plain that the circulating quantity of strong solution will accumulate faster than the weak. The tendency will be for the strong solution to gather in so great excess that it cannot be handled, and for the quantity of weak to fall short of what is required for washing purposes. A balance may be preserved by occasionally pumping up a sump full of strong solution into the weak storage tank, and properly diluting it with water to reduce it to the strength of a working weak solution.

The following summary indicates the time taken up by the various stages in cyaniding operations at Standard Plant No. 1, Bodie. These average results are compiled from one of the monthly tables, and cover a treatment of 1,995 tons, during which a series of sizing tests (taken daily) showed only 37% of the tailings fine enough to pass through an 80-mesh screen: Filling vat, 9 hours; leveling off charge and saturating, 3½ hours; preliminary soaking, 4 hours; bringing outgoing solution up to 18% in strength, 20 hours; soaking with strong solution, 20 hours; displacing strong with weak, 29 hours; displacing weak with water, 13 hours; sluicing, 3 hours—total, 101½ hours.

I have, in the foregoing, attempted to outline a system of applying and manipulating cyanide solutions, which, however, is by no means suited to all localities and conditions. It is recommended merely as a working basis, subject to almost infinite variation. The *modus operandi* may be safely said to differ in each mill. Each operator has his own peculiar method or hobby, established, perhaps, after a vast amount of experimenting with the particular class of material he has to deal with. In illustration of this point, I have gathered from various sources interesting data of the different stages in cyanide practice at important works.

M. Eissler* gives the following interesting particulars of the

* "The Cyanide Process," p. 48.

treatment of a 135-ton vat of tailings at the Worcester Works in South Africa. He cites a certain vat No. 4 as an example:

"On the 20th of August, 1894, this vat, which holds 135 tons, had been filled with tailings. It takes about five hours to fill such a tank. One ton of tailings is equal to 27 cu. ft. From 3:15 to 6:20 P. M., 10 tons of alkaline wash were pumped into the vat; at 8:10 P. M., 5 tons of strong solution were pumped on; at 10:45 P. M., 5 tons more; and at 3:30 A. M., 5 tons additional of strong solution pumped on. On August 21, 5 tons of strong solution were pumped on at 9 A. M., 1 P. M., 5:20 P. M., 10:15 P. M., 3:40 A. M. On August 22, 5 tons of strong solution were pumped on at 9:15 A. M., 1:30 P. M., 6:30 P. M., 9:35 P. M., 4 A. M. On the 23d, at 7:30 A. M., 5 tons of strong solution were pumped on, making a total of 70 tons of strong solution. And on the same date, at 1:50 P. M., 7:15 P. M., and 1:25 A. M., respectively, 7 tons of weak solution were pumped on, making a total of 21 tons of weak solution. On the 24th, at 7:30 A. M., 6 tons of wash-water were pumped on, and at 1:50 P. M., 5 tons additional, making a total of 11 tons of wash-water. The tailings were then leached dry and discharged on the morning of the 25th of August."

According to the same authority, the treatment at the Crown Reef Works is as follows: "It takes 30 hours to fill one of these vats (500 tons). No. 3 vat, for instance, holds 550 tons of tailings. Say at 12:30 on the 29th of August, 50 tons of 0.05% of cyanide solution is run on to drive out the water. When the solution is run through, 75 tons of strong solution, 0.3%, are put on. On the 30th of August, 75 tons more of 0.3% are put on. On the 31st of August, 65 tons of 0.15% are put on. From the 1st to the 3d of September, 300 tons of 0.05% solution, inclusive of 25 tons of wash-water, are put on. Altogether the treatment takes six days, the total quantity of solution employed being 565 tons for this particular tank, in which the tailings assayed 5½ dwts., and the residues 1.1 dwts. An extraction of 80% was obtained, which was rather above the average.

At the old Simmer and Jack Works, now used as a slimes plant, the different operations are given as follows: First, caustic wash (water or otherwise); pumping, 3 hours; contact, 1 hour; leaching, 8 hours. Second, strong solution is pumped without stopping the leaching until all is on. Takes about 8 hours. Amount used about 160 tons. Third, weak solution is kept circulating as previously explained, about 40 hours. Fourth, water

wash (if necessary): Leach dry, 24 hours; time it took to load tank, 12 hours; total from the time of starting filling till ready to discharge, 96 hours. Total amount of strong and weak solution between 500 and 600 tons."

At the Waihi Works, New Zealand, the plant consists of 38 22½-ft. circular leaching vats, with accessory tanks, pumps, etc. The following operations are carried on: Filling vat, 30 tons, two men, 2½ hours; strong solution, 7 tons, 0.35% KCy leaching, 30 hours; weak solution, 7 tons 0.1% KCy, with vacuum, 15 hours; first wash water, 6 tons, with vacuum, 24 hours; second wash, 6 tons, with vacuum, 36 hours; discharging vat, one man sluicing, 2 hours; taking up and cleaning filter bottom, 4½ hours—total, 114 hours.

The plant of the Cassel Gold Extracting Co., New Zealand, consists of eight leaching vats, each 22½ ft. in diameter and 8 ft. deep. The mode of operating is as follows: Filling leaching vat, 30 tons, 3 men, 8 hours; preliminary lime or water-wash, 6 tons, with vacuum, 6; leaching strong solution, 8 tons, 0.6% KCy, 30; weak solution, 4 tons, 0.2% KCy (from strong sump), 12; washing, using vacuum, first weak cyanide wash (from weak sump), 4 tons, 12; second, 12; third, 12; fourth, water-wash, 4 tons, 12; discharging vat, one man sluicing, 4—total, 108 hours.

In the Waihi plant ore is treated directly; in the Cassel Co.'s plant (recently acquired by the Waihi Gold and Silver Mining Co.) tailings are treated.

Direct Treatment of Tailings.—The cyanide process was originally applied commercially to the treatment of accumulated mill tailings. Its range has gradually extended, however, to the direct treatment of various classes of ores, and to tailings run directly from the batteries into cyaniding vats.

Accumulations of tailings are rapidly becoming exhausted, and already, in a number of instances, direct treatment has been successfully applied. In South Africa two methods of filling vats are in practice, which are known as the *intermediate* and *direct* filling. In intermediate filling the tailings are discharged into settling or distributing tanks, by means of which the slimes and sands are separated, and the slimes carried off into pits for subsequent treatment. The sands are then transferred to leaching vats, where the cyanide solutions are applied. In direct filling the coarse sands and slimes are separated by means of spitzluten; and the cyanide treatment is carried on in the same vat in which the material is

discharged. The slimes are either carried off into tanks for immediate treatment, or into slime pits.

These methods have been very well described by M. Eissler,* whose account I quote in full:

Intermediate Filling.—"The first attempts at intermediate filling were made by running the battery tailings to the center of a circular vat and allowing the overflow to take place at one point. This did not prove successful, because the sand piled up in a central conical heap, and the slimes settled in the deeper water around the sides of the tank. The next plan was to run the pulp into the vat through a series of stationary launders, delivering at several fixed points. This method improved the distribution, but the result was still unsatisfactory. Then, in order to give a uniform overflow at every point of the periphery of the vat a circular trough was fixed round the top to collect the overflow, and deliver it to a launder.

"Each of these alterations was a step in the right direction, but the system of settling could not be considered successful until after the introduction of an automatic revolving distributor. . . . The distributor is fixed on an iron column in the center of the vat, the bends at the end of the pipes cause the apparatus to revolve by the reaction of the pulp as it leaves the pipes. Each pipe has a different length, in order to distribute over a number of concentric circles. This also has its faults, as it was found that the slimes collected in narrow rings between the outlets of each pipe, giving rings of clean sand alternately with rings of slime. The difficulty was overcome by attaching flattened nozzles to the ends of the pipes, causing the pulp to spread over a wider area, and also by increasing the number of pipes.

"The arrangement is a hemispherical bowl, from which radiate 8, 12 to 16 pieces of pipe of different length; it is set in motion by the centrifugal action of the discharging water, something similar to a garden sprinkler, only revolving slowly. The bowl is covered with a coarse screen, so as to prevent chips or leaves entering and choking the pipes. The diameter of the discharge pipes is 1½ to 2½ in. (This distributor was devised by Messrs. Butters and Mein.)

"It is necessary to fill the vat with clean water before admitting the pulp. If this is not done the water is practically stationary, and a constant settlement of slimes takes place until the vat is full, and the overflow begins, in which case the tailings in the lower

* "Transactions Institute of Mining and Metallurgy," vol. iii., p. 54.

part of the vat will always be more slimy than those in the upper part. For the same reason it is essential that the overflow be continuous until the vat is full of sand, for if any stoppage takes place, slime settlement in excess occurs, and a complete layer of slime is formed across the vat, which prevents the overlying sands from draining dry. Therefore, when the battery is stopped an equal quantity of water should be supplied to the vat. When the pulp is admitted into the tank previously filled with water, the light slime remains in suspension and overflows into the annular ring which surrounds the tank at the top and from the discharge opening is carried by a launder to the slime pits.

"When the vat is filled with tailings the outlet pipe below the filter is opened and the water allowed to drain off. The draining takes about 15 to 24 hours. When holes are dug down to the discharge doors, water again commences to flow from the outlet, consequently it is advantageous to dig these holes about six hours before discharging.

"One very important matter is the proper size of vat to be used for a given tonnage crushed in the battery. It is of course desirable to catch as large a quantity of slimes with the sands in the tailings as is possible without rendering the product unleachable. When the vats are too small they carry away too much fine sand with the slime, and if they are too large they catch too much slime, which settles in excess. The great difficulty to be overcome yet with these intermediate vats is to get the last foot or two near the bottom properly drained. If the tailings are discharged and transferred to the leaching tanks in this wet condition, the excess of moisture dilutes the cyanide solution.

"To facilitate and hasten the leaching various devices have been adopted. At the Princess Works, where the ground is steep, the drainage pipe has been extended down to the reservoir, thereby causing a natural suction. At the Simmer and Jack Works the drainage pipe is connected with a steam exhaust, acting like an ejector—causing a vacuum below the filter, and thereby the rate of leaching is increased.

"At the Worcester Works the vats catch from the crushed ore from 75 to 80% of good leachable tailings, containing 12% moisture, after draining 18 to 24 hours.

"The following are the sizes of the intermediate vats erected at some of the works:

"Meyer and Charlton G. M. Co., treating 120 tons per day, has four vats, each 20 ft. in diameter and 8-ft. staves.

"Pioneer G. M. Co., treating 70 tons daily, has two vats, each 20 ft. in diameter and 14-ft. staves.

"Worcester G. M. Co., treating 70 tons daily, has two vats, each 20 ft. in diameter and 8-ft. staves.

"Princess G. M. Co., treating 85 tons daily, has two vats, each 20 ft. in diameter and 7-ft. staves.

"The Robinson G. M. Co., treating 330 tons daily, has six vats, each 24 ft. in diameter and 11-ft. staves.

"It will be seen from the above that the Meyer and Charlton crushes nearly twice as much as the Worcester company, but the collecting vats are the same size, consequently the amount of tailings retained is considerably less. When all the pulp is running into one vat only about 66% of the crushed ore is caught, but the whole of this is clean sand and drains sufficiently. If, however, the total pulp from the battery were run into two vats, about 80% of the crushed ore, instead of 66%, would be obtained. From the distributing tank, after the water has been leached out, the ore is discharged through bottom dischargers into trucks and taken to the leaching tanks. In some localities the distributing tanks are on a higher level than the leaching tanks, and the trucks are then run by gravitation to the leaching tanks. At some works the distributing tanks are at a lower level than the leaching tanks, and then the trucks have to be hauled up by steam power; for instance, at the Simmer and Jack.*

"The framework of the tram lines on which the trucks are hauled up to the leaching tanks rest inside the tanks and on the masonry foundation, and at large works there is generally a double line of rails on the top of the tanks. The vats and storage tanks are in the open, and not covered by a building."

Advantages of Intermediate Filling.—(1) It is claimed that by means of Butters' distributor from 75 to 80% of sands, both coarse and fine, with some slimes, are collected in the intermediate tanks, the bulk of the slimes escaping with the effluent water being practically free from sands.

(2) The water is drained off as far as possible, and when the

* The latest and best practice in South Africa is the erection of the intermediate vats directly above the leaching vats; *i. e.*, superimposed upon the leaching vats. This does away with hauling the wet tailings, and insures a considerable saving in cost of treatment. A plant for the direct treatment of tailings is now (August, 1898) being constructed upon this plan at Bodie. (See Illustration)

intermediate vat is discharged through the bottom dischargers, the sands during the operation get thoroughly mixed up, thus being in the best condition for treatment by cyanide.

(3) Oxidation of pyrites is very slight, so that very little cyanide will be consumed.

To an impartial observer it would appear that the system of intermediate filling would commend itself as the one which is more practical, as the tailings undergo, so to say, a special preparation for the subsequent lixiviation. The expense of transferring the tailings from the intermediate tank to the leaching tank is so slight that it cannot be considered as an important item.

The cost of charging tailings and discharging the residue has been brought down at the Robinson Mine to 7d. per ton of 2,000 lb.; it generally stands in the accounts of other works at about 1s.

Direct Filling.—This method, introduced at the Heriot, the City and Suburban, Crown Reef, Paarl Central and Geldenhuis Estate G. M. companies, consists in passing the pulp leaving the plates into a hydraulic separator, a kind of crude spitzlutte. The pulp is divided into two streams, one overflowing, carrying slimes with the very fine sands, the other consisting of coarse sands, some fine sands and slimes, which are conveyed by means of an india rubber hose to the leaching tanks, in which one or more Kaffirs are employed to effect the even distribution of the pulp, by moving the hose about to different parts of the vat. The water passes off by adjustable gates fitted inside the vats, carrying with it fine sands, slimes, and some coarse sands.

The advantages of this process are:

(1) This method treats pyritic tailings with the minimum of oxidation as they are not exposed to the action of the air from the time they leave the battery.

(2) A second handling of the tailings before treatment is avoided.

(3) A preliminary rough concentration, or rather classification of the coarser particles of the tailings is effected.

There is at present a great controversy going on regarding the advantages of direct filling, as against intermediate filling.

Mr. Bettel points out that the disadvantages of the direct process are:

"(1) The tailings pack tightly in the vat, and consequently do not drain completely, and a diffusion of the first cyanide solution which is applied takes place at the commencement of leaching, causing loss of cyanide and gold.

PLATE VII.

New Works of Standard Company in Process of Construction at Bodie for Direct Treatment of Tailings.

Showing double tier of leaching vats. Tailings will be run into upper or intermediate vats by means of Butters' distributors, subjected to a short treatment, and discharged into lower vats through circular bottom-discharge doors.

"(2) The distribution of the sands and slimes is not so even, and some sands escape treatment, being protected by impervious layers of slime, the cyanide naturally escaping by the paths of least resistance. In leaching tanks where an uneven distribution of slimes and sands takes place, the slimy portion will not drain off, and on discharging such a tank it is easily noticed that the streaks of slime are saturated with moisture, and are still gold bearing, whereas the sandy portion has the solution drained off. The importance of an even distribution and mixture of the pulp can hardly be estimated.

"(3) At most of the works where direct filling is introduced square cement tanks are employed, and the discharging of these is not so practical as with wooden ones fitted with bottom discharges."

The disadvantages of direct filling do not appear to lie in any defect of the principle itself, but rather in the imperfection of the method, as applied. The system certainly has one very important advantage—it does away with the additional expense of handling the tailings a second time. This may not be an important item in South Africa, but in the United States and elsewhere it is certainly worth considering. The additional time consumed in transferring material from intermediate vats to leaching vats is also a matter of importance. The well-known difficulties in the way of direct filling have already been alluded to. There seems to be no perfectly satisfactory method of mixing uniformly the slimes and sand, and of admitting to the vat exactly the proper amount of slimes; and so long as the case remains as it is, intermediate filling will probably have the preference. A method recently advocated in South Africa is to have two vats charging alternately from the separator. As soon as one vat is filled to a certain depth (1 or 2 ft.) the stream is turned into the second vat. The slimes are then allowed to settle for a time in the first vat, and drained off. The outlet under the filter is opened, the moisture in the sand partly drained off, and the coarse material (covered by a thin layer of slimes) is shoveled or raked over, so that the sand and slime are intimately mixed. The stream is then turned back to No. 1 vat, and No. 2 vat treated as I have described. This process is continued until the vats are filled. The method seems feasible. The cost of labor should not be nearly so much as for transferring the pulp to other vats; and one great objection to direct filling—a lack of oxygenation—is thus eliminated.

The question to be solved is whether the increased extraction

obtained by intermediate filling (from aeration of the pulp, and a more even distribution of sand and slime) offsets the additional cost of labor. In some localities it might; in some it might not. Only exhaustive experiments and calculations on cost of treatment and extraction can demonstrate which method is the best.

Treatment of Ores.—In the cyanide process as applied to ore the principles are the same as in the treatment of tailings; the only marked variations being in certain methods of preparing the ores, and in the mechanical arrangements for transferring it from the ore bins to the leaching vats. At Mercur, the ore being very porous, is delivered in a coarse state to the vats; in Colorado it is crushed to varying degrees of fineness, according to the readiness with which it yields up its gold, and its permeability to cyanide solutions. Some material requires no preliminary treatment beyond crushing to suitable fineness; other materials require roasting.

"The size of ore leached varies from through a ¾-in. mesh at the Marion (Utah) to through a 40-mesh at the Metallic Reduction Co.'s mill at Florence (Colorado); and the crushing and sizing machinery ranges from the single Gates crusher and one trommel at the former, to three Gates crushers, three multiple-jaw crushers, six sets of rolls, and numerous trommels at the latter."

"The method of conveying ore from pulp-bin to leaching vats is almost universally by cars on a track over the vats. The Commercial Mill at Bingham, Utah, has inclined spouts, through which the ore flows from a centrally located bin into the tanks, and a few mills have bins directly over the tanks."

The methods of applying solutions appear to be quite as various in cyaniding ores as in treating tailings.

"The preliminary treatment includes the use of lime, caustic soda and sodium dioxide. Lime, when used, is mixed directly with the ore. In some cases the pulp is then washed with water until the lime is all washed out. At other mills the solution is put on at once. Caustic soda is used in the same way, and also in solution as a preliminary wash. . . . Many mills begin the leach by admitting the solution at the bottom of the vat until the ore is covered. The solution is then turned on top and allowed to run on at the top and drain off at the bottom simultaneously for a certain number of hours, the surface of the ore being kept covered; very good results are obtained from this mode of treatment when the tanks are allowed to stand for a short time after the pulp is covered before the drainage valve is opened and the solution turned

on top. This allows the whole mass to become thoroughly saturated, and the slow draining prevents the formation of channels. Any great depth of solution on top of the charge causes it to 'pack,' and an uneven extraction follows. In the Mercur district the ore is covered with solution, which is allowed to stand from 30 minutes to 6 hours and then drawn off. This operation is repeated from 8 to 35 times. Here the material leached is so coarse that there is no danger of 'packing.' Each operation of covering takes from 2 to 6 hours. A few mills cover the pulp with solution, allow it to stand 48 to 96 hours, draw it off and wash.

Many of the mills follow the strong solution with a wash of weak solution (0.1% or less). This is in turn followed by a water-wash, which flows through the zinc-boxes into the weak-solution tank, and becomes the first wash for the next charge.*

Methods of Standardizing Sump Solutions.—After passing through the zinc-boxes, the strong solutions, upon titrating with silver nitrate will be found to have deteriorated considerably in strength in passing through the various stages in the process. To obtain the best results, it is necessary to raise this solution to the accepted standard strength, before using it on a fresh charge of tailings.

Various methods of standardizing are used in South Africa and elsewhere. Liquors may be raised to normal strength in the sump by placing the additional amount of cyanide required in the last compartment of a zinc-box, and allowing the outflowing solution to dissolve it; or by placing it in a perforated tray, so suspended by means of pulley and rope that it can be immersed in the sump. Another method is to prepare a strong solution of cyanide in a separate tank, or "cyanide dissolver," and to draw upon this in raising the storage-tank liquors to standard strength. Where there are storage tanks, the custom of standardizing in the sump is not a good practice. If the sump is full, and only a portion of its contents can be accommodated in the storage tank, it is obviously impossible to raise only a part of the sump liquors to standard strength without raising all. Where there is no storage tank, of course all the solution in the sump must be kept standardized.

The Cyanide Dissolver.—The system of strengthening the storage tank solution with cyanide already dissolved is open to the objection that it requires a separate dissolving tank, and involves some inconvenience, not alone in the dissolving, but in the subsequent cal-

* These extracts are quoted from Packard's article, "The Cyanide Process in the United States," "Transactions of American Institute of Mining Engineers," vol. xxvi., p. 709.

culations of the necessary amount of cyanide to add in standardizing. It is the only available method, however, where impure commercial cyanide is used, containing the black carbide of iron and other insoluble matters, which, if allowed to pass into circulation, interfere with the efficiency of the solution. The commercial cyanide formerly used in cyaniding works contained a variable amount of the available salt; consequently it was necessary to calculate the percentage of impurity present before standardizing. A separate dissolving tank containing the strong cyanide solution at a uniform strength was therefore a necessity. It served the double purpose of a storage tank for a reliable standardizing solution, and of a settler for the insoluble impurities in the cyanide.

At the present time a very clean and very pure article is obtainable in the American market, whose strength of available cyanide (98 to 99%) can be depended upon; and which can be directly mixed with the stock solutions without contaminating them.

Where such cyanide can be had, a good method of standardizing is to suspend the necessary quantity in a bag directly under the discharge into the storage tank, and allow the incoming solution to dissolve it; or better, place the cyanide in a tin or wooden box, with bottom and sides perforated, and fixed permanently on a shelf or bracket directly under the discharge. This method will be found to insure a perfect mixture of the cyanide with the solution to be standardized.

To Standardize with Pure Cyanide.—The storage tank should be provided with a gauge or telltale constructed similar to that on the gold tanks, registering the depth of solution within. The amount of solution to be strengthened should, for sake of convenience, always be considered in terms of feet, as indicated on the telltale, and not in pounds or gallons; and in each plant, the number of pounds of cyanide necessary to make one foot of standard solution in the storage tank should be calculated, and this will remain a fixed figure in the subsequent calculations. For example, in a tank 12 ft. in diameter (calculating closely enough for practical purposes) there are 3½ tons of solution to the vertical foot. If we assume the standard solution to contain 0.2% KCy, a ton will contain 4 lb., and 3½ tons, or 1 ft. of solution, 14 lb.

Formula for Standardizing.—The amount of cyanide to be added each time sump solution is pumped up to the storage tank may be calculated by the following formula:

Let A equal number of pounds KCy in 1 ft. standard solution;

B, strength of sump solution; C, strength of standard solution; D, number of feet of sump solution to be standardized; X, amount of KCy to be added.

$$\text{Then } X = A - (A \times \frac{B}{C}) \times D.$$

For example, if our standard solution is 0.25% in strength, and we wish to raise to standard strength $2\frac{1}{2}$ ft. of sump solution at 0.21% strength, the number of pounds KCy to be added =

$$17\tfrac{1}{2} - (17\tfrac{1}{2} \times \frac{.21}{.25}) \times 2\tfrac{1}{2} \text{ or}$$

$$2.8 \times 2\tfrac{1}{2} = 7 \text{ lb.}$$

To avoid the trouble of making this calculation every time strong solution is pumped up, a table may at once be devised which at a glance will indicate the amount of cyanide necessary in each case.

STANDARDIZING TABLE (ASSUMING STANDARD STRENGTH TO BE 0.25% KCy).

Feet....	½	1	1½	2	2½	3	3½	4	4½	5
0.17%....	2.8	5.6	8.4	11.2	14	16.8	19.6	22.4	25.2	28

In this table the first row of figures indicates the number of feet (from ½ to 5) of sump solution to be standardized. The second line indicates the different amounts of cyanide to be added. Assuming standard strength to be 0.25%, suppose we wish to raise $3\frac{1}{2}$ ft. of sump solution at 0.17% to 0.25%. Find $3\frac{1}{2}$ in the first line, and read the figure underneath; thus 19.6 lb. is the quantity to be added. Similarly, the other columns, from 0.18 to 0.25% may be figured out.

Such a table may be made out for any standard strength of solution and posted up in a convenient place near the storage tank.

As soon as the amount of cyanide to be used is determined, it should be put in the dissolving box and pumping commenced. No solution should be drawn from the storage tank until the whole quantity to be standardized has been raised from the sump.

CHAPTER IX.

PRECIPITATION BY ZINC.

THE strong and weak solutions leaving the leaching vats are gathered in their respective gold tanks and thence passed through the zinc-boxes into the sumps. From the weak sump the solution is pumped up to the weak storage tank to be re-used as a wash for strong solution; the solution from the strong sump is raised to the strong storage tank, built up to standard strength, and again circulated. The methods of standardizing this solution will be discussed in the next chapter. In the meantime we will take up events in their proper sequence and consider the second stage in cyaniding operations, the recovery of the precious metals from gold-bearing solutions. This is accomplished by means of zinc shavings in the zinc or extractor-boxes.

The number and dimensions of zinc-boxes suitable to mills of different capacities are given in the table (p. 40). The arrangement of the boxes is shown in the plate.

The use of filiform zinc shavings as a precipitant is generally adopted wherever the MacArthur-Forrest process is in operation. "The zinc used for precipitation purposes should be the best quality found in commerce, and should not contain arsenic or antimony; a small percentage of lead, however, does no harm, but rather tends to promote rapid action by forming a voltaic couple with the zinc" (MacArthur).

Zinc-sheets, zinc-amalgam, zinc-dust, and zinc-fume have been variously tried, but discarded in favor of the shavings. Sheet-zinc offers too little surface for precipitation; and by reason of its smooth surface preventing the free escape of hydrogen gas, retards the electrolytic action of the gold-zinc couple.

Various Forms of Precipitant.—Zinc-dust and zinc-filings are not sufficiently permeable to accommodate the necessarily rapid flow of solutions through the boxes. The use of zinc-fume (a finely

divided by-product of zinc distillation) was recently advocated as a more perfect precipitant than the shavings, and as a means of dispensing altogether with the use of zinc-boxes. The method of using it is to agitate a certain quantity of it with gold-charged solutions, allowing it to settle, and draining off the supernatant liquor. I know of no instance where zinc-fume is being successfully used on a large scale. The great objection to it would seem to be the labor involved in mixing it with, and separating it from, solutions, a very great labor in comparison with the work required in attending to zinc-boxes. Sulman claims that the precipitation obtained with zinc-fume is much better than that usually obtained with shavings. In precipitating gold from weak solutions the zinc-fume might possibly act better, in certain instances, than the shavings. The writer speaks of having devised an apparatus by means of which the fume may be automatically applied to a continuous flow of liquors from the leaching vats; but the apparatus is not described.*

Zinc shavings possess the distinct advantage of allowing the free and rapid passage of cyanide solutions through them, and of not clogging the screens through which the gold precipitate falls.

The excellence of zinc in this form as a means of precipitation was thought to depend upon the vast extent of surface exposed to the action of solutions. Professor Christy,† however, ingeniously contends that shavings act better than smooth zinc sheets by reason of their presenting an infinite number of points or ragged edges, which favor the escape of hydrogen, and so accelerate the electrolytic action.

Method of Preparing Zinc Shavings.—The method generally employed of preparing zinc shavings is to cut them from a solid cylinder of zinc disks, mounted on the mandril of a lathe. The disks, 12 in. in diameter, are punched in the center with a 1-in. hole, through which the iron mandril runs, the series of disks being held together by means of a system of iron plates and nuts. The shavings are cut by hand by means of a chisel supported on a rest.

This system is objectionable on account of the excess of cost of labor over other methods. The shavings may be cut automatically with a minimum of attention on a machine lathe, a method employed at Mercur, Bodie, and elsewhere.

At the Standard Works in Bodie the zinc is cut on a compound-

* "Transactions Institution of Mining and Metallurgy," vol. iii., p. 223.

† "Transactions of American Institute of Mining Engineers," vol. xxvi., p. 762.

geared machine lathe, in the machine-shop connected with the stamp-mill. The lathe is provided with a wooden mandril 3 ft. long and 9 in. in diameter. Around this mandril are tightly wound five sheets of commercial zinc, No. 9 gauge, 3 ft. wide and 7 ft. long, the edge of each sheet being soldered to the layer underneath. The side-cutting tool of mushet steel is fixed to the carriage and adjusted to the edge of the core of zinc sheets. The mandril is geared to 84 revolutions per minute, or 242 revolutions to 1 of the carriage-screw. Thus, in 15 minutes the carriage moves 1 in. and the mandril makes 1,260 revolutions. The shavings cut are $\frac{1}{1260}$ in. in thickness and about $\frac{1}{32}$ in. wide. Five sheets of zinc of dimensions given above weigh about 65 lb.; hence, to cut this amount into shavings requires about 9 hours. Such a lathe, working automatically $4\frac{1}{2}$ hours per day, will cut enough shavings to supply a 75-ton plant, operating under average conditions. As the zinc shavings run off from the lathe they are caught between a series of small iron rolls, operated from the mandril, by which they are guided into a box under the lathe provided for the purpose (Fig. 15). If the shavings are allowed to wind around the mandril they must periodically be pulled off in bunches, which have to be disentangled before the material can be used for precipitation. Before the system of rolls was devised a man was kept busy several hours a day preparing the shavings.

An assistant in the machine-shop winds the sheets on the mandril and keeps the cutting-tools sharpened; but apart from this the lathe requires very little attention.

Shavings cut to the dimensions given will expose per lb. about 1,530 sq. ft. of surface. A sheet of zinc exposing $42\frac{1}{2}$ sq. ft. of surface will expose, when cut up into shavings, over 400 times as much.

Management of Zinc-Boxes.—All the compartments in a zinc-box may be filled with shavings except the last, which it is advisable to leave as a sort of settler for any particles of precipitate carried over from the other compartments. In a well-attended precipitation box, however, there is very little likelihood of any of the fine gold slimes being carried down so far, unless too much work is imposed upon the box. The precaution may be taken of placing a screen of burlap on the top of this compartment, to prevent any possible escape of precipitated gold into the sump. At the Standard Works a dark precipitate gathers on the burlap screens which was at first supposed to consist of floating particles of gold slimes.

Upon assaying, however, it proved of no significant value, and was probably only one of the many complex precipitates formed during leaching.

The zinc should be laid in loosely, care being taken that the filaments are not bunched, but well separated. The shavings should be uniformly distributed, and the corners of the compartment well filled to prevent the solution rising in channels.

The flow of solution through the boxes should be as far as possible regulated to a uniform speed. In a 75-ton plant, treating ores having an average leaching rate, three zinc-boxes, each with an average flow of 1 ton (240 gal.) per hour, will accommodate all the strong solution in circulation. During the 24 hours it will occasionally be found necessary to regulate the flow according to the height of solution in the gold tank, the discharge into the gold tank not being perfectly uniform. A telltale, properly marked off in inches and feet, with an indicator regulated by a float inside the tank, should be attached to the side of each gold tank to indicate the height of solution.

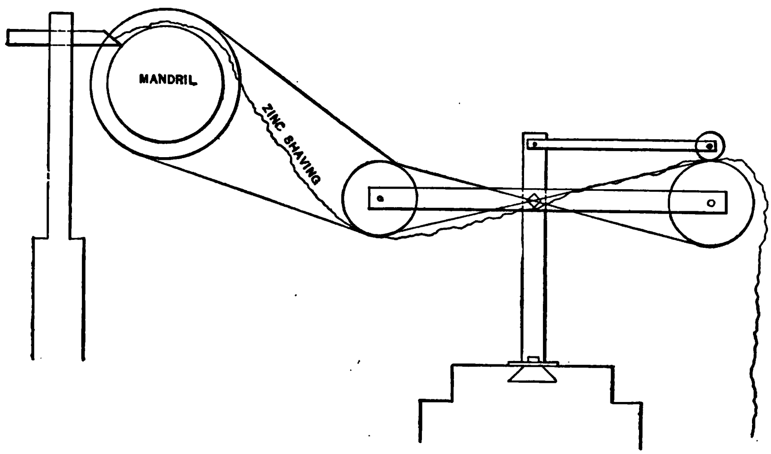


FIG. 15.—DEVICE FOR GUIDING ZINC SHAVINGS FROM THE LATHE.

In some cases, where the precipitation is imperfect and diffused through all the compartments, a slower rate of flow will be required in such instances it may be necessary to use a fourth box, or increase the length of the original boxes.

Where good precipitation takes place a solution containing 5 or 6 gm. of gold per ton ought to produce in an hour or two a dis-

tinct gravish-black discoloration on the zinc in the first compartments. The discoloration should grow fainter toward the end compartment, and in the last two or three ought to preserve its luster for 24 hours or more. The condition of the last compartment will indicate to the practiced eye whether the box is doing its work properly, a rapid discoloration being usually coincident with an imperfect precipitation at the head. In some instances, however, where solutions contain much organic matter and are accompanied by frothing in the compartments, the precipitation will be unavoidably diffuse, without necessarily being poor. These conditions can only be positively determined by frequent assaying of inflowing and outflowing solutions. A good remedy for diffuse precipitation is to increase the quantity of lime, lime having the effect of clarifying cyanide liquors from organic matter in solution.

Color of Gold Deposit on Zinc.—In normal zinc precipitation the metallic deposit on the zinc is a brownish-black, or grayish-black. In imperfect precipitation it is frequently gray or a dull metallic hue; a condition which may be traced to a variety of causes. The use of caustic soda is apt to produce a grayish deposit of ferro-cyanide of zinc in the boxes, but the color is usually due to the presence of organic matter, or to acids and soluble salts of iron improperly neutralized. This variation in color between gray and black, dependent upon some obscure chemical changes by no means understood, is the most puzzling feature of zinc precipitation.

At the Standard Works the practice was at one time followed of saturating a vat with weak solution of about 0.06% KCy, and displacing it with the strong. This weak solution being drained off and passed through the weak zinc-boxes, produced a pale-gray precipitate on the zinc very different from that occurring in the strong boxes. Later on the process was reversed in practice; a strong solution was used for saturating purposes and a weak to displace the strong. The conditions, strange to say, were not reversed during precipitation, but a very perfect dark precipitation produced in both strong and weak boxes. A comparison of results eliminated one possible cause for gray precipitation, namely, a poverty of the solution in gold value, since frequent assays revealed a high gold value in the weak solution first flowing from the leaching vat.

The conclusion reached was that there was a strong selective action going on in the zinc-boxes, determined wholly by the strength of the solution in cyanide. Other substances besides gold and silver would appear to have an active tendency to precipitate on zinc, a

tendency which is only overcome by the proper amount of cyanide in solution. For instance, if there is a preponderance of these precipitable substances over the gold present, only a solution of certain strength will enable the gold to deposit first, to their exclusion; if the solution be weak in cyanide they will appear on the zinc as a deposit of dull, silvery-gray color. This hypothesis seems to be substantiated by the fact that we often find gray precipitation in the upper end of a box, and black or gold precipitation in the lower compartments. It is a well-known fact that copper will precipitate from cyanide liquors to the almost total exclusion of the gold, giving the zinc a coppery luster. This has been observed in the cyaniding of tailings which have been treated with copper sulphate in certain pan-amalgamation processes.

The phenomena observed at the Standard Cyanide Works may, on this supposition, be explained thus: Bodie tailings contain a certain amount of the acid products of pyritic decomposition. A weak solution slighly charged with combinations of these soluble salts with cyanide of potassium, owing to an imperfect neutralization with lime, precipitated these compounds on the zinc in preference to the gold. When the practice was reversed, and strong solution used to carry off the excess of soluble salts, precipitation of the gold was more perfect, simply because the solution was stronger in cyanide. The weak solution, run through the boxes at a later stage of the leaching process, produced a black precipitation, because by that time either the solution had become clarified, or the foreign compounds had been rendered innocuous by one of those complex chemical changes so little understood.

A poor precipitation has in some instances been corrected by an addition of cyanide to solutions entering the zinc-boxes. This would seem to be the remedy for gray precipitation as well, although I do not know of its ever having been applied in practice to correct this particular trouble. In some cases, as one might naturally expect, the gray coloration on the zinc may be corrected by an increase in the quantity of lime; in other instances, however, where the gray color is undoubtedly due to the precipitation on the zinc of certain obscure compounds whose action cannot be neutralized by caustic alkalies, the only remedy would appear to be in strengthening the cyanide solutions before passing it through the boxes.

Care of Zinc-Boxes.—The boxes should be frequently examined and the zinc re-arranged at least once a day. While the zinc is

being shifted, the solution should be turned off and the flow through the box checked until the precipitate has had time to settle.

As precipitation goes on, the zinc darkens in color, and the shavings in the first compartment lose their wiry consistency and become soft and stringy. When the shavings have gathered more of the precipitate than they can hold, the latter drops off and falls through the screen into the bottom of the compartment; the zinc gradually decomposes, and either floats in the solution or subsides into a soft mass on top of the screen.

As the shavings in the first compartment settle down it is usual to replenish them with zinc from the end of the box, as zinc upon which gold has already precipitated is more readily attacked than fresh shavings. The end compartments thus emptied may then be filled with new material. The custom of shifting the contents of the compartments, that is, of moving the contents of each compartment to the one ahead, is to be condemned on account of the risk of losing precipitate from solution thus agitated in the handling.

Weak Solution-Boxes.—The difficulty of precipitating gold from weak solutions usually necessitates the use of a longer column of zinc than for strong solutions. A series consisting of two 9-compartment boxes will, under favorable conditions, accommodate all the weak solution in a 75-ton plant. These boxes should be so connected alongside of each other that the solution, after passing through the first, enters the second, and so comes in contact with 18 compartments full of zinc shavings. This arrangement might have to be modified in some instances. Three or four boxes might be required for weak solution, according to the difficulty experienced in depositing the gold. If it were not for the space taken up a long box might at once be constructed containing the desired number of compartments. In cases where weak solutions will not deposit gold on zinc, the weak solution is simply used as washes, and the surplus run to waste or discharged with the spent tailings, no wash-water being used. This, however, necessitates a considerable loss. (See Siemens-Halske Process.)

Proper Degree of Precipitation.—In good precipitation a very large proportion of the gold and silver (between 95 and 99%) should be deposited on the zinc. In many plants operating successfully the percentage is considerably less than this, as much as 2 or 3 gm. of gold per ton of solution being discharged into the sump. In one of the Bodie plants three boxes of 10 compartments each are

used for strong solution, and two series of boxes (18 compartments to a series) for weak.

The following table of assays made at random on solution-samples from various zinc-boxes at different intervals in the season's run indicates a high percentage of precipitation in both strong and weak-solution boxes. These results were obtained by evaporating 235 c.c. (about 8 assay-tons) of the solution in each case, and assaying the residues.

	Strong or Weak.	Value of the Incoming Solution per Ton.			Value of the Outgoing Solution per Ton.		
		Au.	Ag.	Total.	Au.	Ag.	Total.
1	Strong (0.14 to 0.16 per cent.)......	$4.24	$0.45	$4.69	$0.05	$0.02	$0.07
2	Strong (0.14 to 0.16 per cent.)......	4.39	.43	4.82	.10	.03	.13
3	Strong (0.14 to 0.16 per cent.)......	4.03	.44	4.47	Trace.	....	....
4	Strong (0 14 to 0.16 per cent.)......	4.13	.46	4.59	Trace.	....	....
5	Strong (0.14 to 0.16 per cent.)......	3.32	.38	3.70	.10	.06	.16
6	Strong (0.14 to 0.16 per cent.)......	3.05	.39	3.44	.26	.07	.33
7	Weak (0.04 to 0.07 per cent.)........	1.24	.14	1.38	Trace.	....	....
8	Weak (0.04 to 0.07 per cent.)........	2.90	.45	3.35	.07	.07	.14
*9	Weak (0.04 to 0.07 per cent.)........	3.82	.57	4.39	.25	.07	.32
*10	Weak (0.04 to 0.07 per cent.)........	4.13	.64	4.77	.21	.08	.29
*11	Weak (0.04 to 0.07 per cent.)........	1.14	.16	1.30	Trace.	....	....
*12	Weak (0.04 to 0.07 per cent.)........	.52	.15	.67	Trace.	....	....

* Assays Nos. 9 and 10 were made on solutions taken from a box containing a considerable amount of aluminous slime. The slightly lower percentage of precipitation is apparent. The two following assays (Nos. 11 and 12) were made on a low-grade solution flowing through the same box, after a complete elimination of the aluminous material, and indicate a normal precipitation.

The progress of precipitation, determined on three different occasions, from strong solution-boxes, at Standard plant No. 1, is given below in tabulated form:

	Box Sampled Just Before Clean-up.			Box Sampled Few Days Before Clean-up.			Box Sampled Just After Clean-up.		
	Gold	Silver	Total	Gold	Silver	Total	Gold	Silver	Total
Value per ton of solution entering first compartment..........	$4.96	$0.38	$5.34	$2.68	$0.28	$2.96	$5.37	$0.46	$5.83
Leaving first compartment........	4.75	.35	5.10	2.48	.27	2.75	3.92	.32	4.24
Leaving second compartment....	4.54	.29	4.83	2.06	.24	2.30	1.86	.29	2.15
Leaving third compartment......	4.54	.28	4.82	1.86	.22	2.08	1.24	.17	1.41
Leaving fourth compartment.....	4.34	.25	4.59	1.24	.12	1.36	1.03	.13	1.16
Leaving sixth compartment......	4.24	.20	4.44	1.03	.08	1.11	.41	.07	.48
Leaving ninth compartment......	1.86	.15	2.01	.20	.04	.24	Trace	Trace	Trace
Percentage precipitated in first four compartments............	12%	34%	14%	54%	57%	54%	80%	71%	80%
Percentage precipitated in last five compartments.............	50%	26%	48%	38%	28%	38%	19%	28%	19%
Total precipitation................	62%	60%	62%	92%	85%	92%	99%	99%	99%

It may be noted here that the precipitation was poorest just before the clean-up, when the first compartments were charged with precipitate and soft zinc. In series No. 1 nearly the whole work of precipitation falls on the last five compartments. This is reversed in the second and third series, noticeably in the third, where 80% of the gold and silver is left in the first four compartments.

CHAPTER X.

CLEANING UP AND REFINING THE PRECIPITATE.

THE difficulties incident to cleaning up and reducing the zinc-gold slimes, or precipitates, are popularly thought to be the weakest points in the cyanide process. It is very true that problems have arisen in connection with this very complex product of zinc precipitation. It is also true that the methods of cleaning up in the best-managed plants have been vastly improved since the first experimental years of operation. At the same time a great diversity still exists in the clean-up practice in different mills. This is due somewhat to the different conditions obtaining in each case; but more, perhaps, to a lack of information obtainable on the best methods employed in practice. As more becomes known of these methods it is safe to predict that the tendency will be toward the acceptance of one simple and efficient system, which with little modification can be employed in all cases.

The desideratum at the present time seems to be a system of handling conveniently the great bulk of raw material taken from the zinc-boxes, and of reducing the refined precipitate to bullion without losses. In many plants the first step in the clean-up process (conveying the zinc slimes from the boxes to the proper reducing tanks and filters) will be found to be a somewhat sloppy operation, unsystematized and awkwardly managed. Buckets are used, from which part of the contents are frequently spilled over on to floors improperly adapted to the easy recovery of such losses; and the various buckets and tubs are not always kept scrupulously clean. Some plants are erected without any provision for cleaning up. Makeshifts are used, which render the operation a clumsy and dirty one; and thus losses undoubtedly occur which contribute to make the whole plant a failure. There is really no reason why a cyanide mill clean-up, if properly managed, should not be as easily conducted and attended by as few losses as a clean-up in a quartz mill, or in any metallurgical process.

Opinions seem to differ a good deal as to the expediency of taking only the finest precipitates from the boxes at the clean-up (such as will pass through a 40 or 50-mesh screen), or of including the small fragments of zinc usually found mixed up with the slimes.

The usual method is to separate from the contents of the zinc-box only the finest of the slimes, to wash the coarser material, and replace the latter in the compartments. These fine slimes are separated out by means of screens (40 or 50-mesh) placed at the end of the side-launder. Such a system involves a considerably less consumption of acid and insures a higher grade of bullion. But it is usually desirable to get as large an output as possible from a mill each month, especially at the commencement of operations, and to make the bullion yield and the yield indicated by assays correspond as closely as possible. It is a well-recognized fact that even after all the slimes are removed from a box, and the balance of the zinc washed as thoroughly as possible, there still remains behind a considerable value, which can be recovered only by destroying the whole bulk of zinc. This value was found to be, in Bodie, between 30 and 75c. per oz. of zinc. It is evident, therefore that to obtain a close correspondence between the actual and indicated yield, something more than the mere precipitates must be taken. It seems a perfectly justifiable practice to remove not only the precipitates, but the mass of short zinc, and leave nothing in the box but the longer shavings.

During the various manipulations of the zinc, in the interval between clean-ups, the brittle, partly decomposed shavings lying immediately on top of the screen become disintegrated into small fragments (short zinc), which fall through the screen and get mixed up with the slimes. These fragments appear to be practically inert so far as their availability for further precipitation is concerned. A mass of this material, which had been thoroughly rubbed and washed, and still retained a value of 30c. per oz., was exposed for several days in a screen to the action of strong solution flowing through a zinc-box, and was found to have been practically unaltered. No precipitation was perceptible, although in the meantime a very active precipitation was going on in the coarse shavings of an adjoining compartment. Such fragments, however, if exposed in a percolator to the action of a slowly percolating solution, will precipitate a considerable percentage of the gold.

Short zinc, if replaced in the boxes after a clean-up (a method which prevails in many large plants) is an actual hindrance, as it

very soon works its way down through the screens and obstructs the flow of solution.

To summarize, the short zinc should not be replaced in the boxes for the following reasons:

(1) It carries a considerable value which it is usually desirable to obtain at the clean-up.

(2) It is practically worthless as a precipitant, being too finely divided to permit of a thorough contact with solutions.

(3) It falls through the screens and obstructs the flow of solutions.

Value Left After a Clean-up.—After an ordinary clean-up in a plant treating 2,000 or 3,000 tons of average grade tailings per month, it is safe to assume that from $1,500 to $2,500 in value remains in the boxes in the form of a film of precipitate on the long zinc shavings. This fact might be easily overlooked in accounting for the shortage generally noticed in the first yield of a new plant.

Two Methods of Treating Zinc Precipitates.—There are two methods in practice of treating the zinc-box precipitates—by roasting and by reduction with sulphuric acid. In the former the zinc residues are destroyed by oxidation; in the latter by conversion into hydrogen and zinc sulphate. Roasting has been the prevailing method in South Africa and New Zealand; but at the present time there appears to be a general reaction of opinion in favor of the acid method. The latter has been extensively adopted in America. Each method has its advantages. Calcination is in most instances cheaper; but the acid method is supposed to give a cleaner and higher grade bullion. On the other hand considerable losses during reduction have been attributed to both methods: to calcination during the rapid oxidation of the zinc, and to the acid treatment "by the formation of a regulus in melting, if sulphates have remained in the slimes by fault of imperfect washing."

At the present time the chief objection to roasting is the unaccountable losses of gold during the operation; while in acid reduction losses appear to have been reduced to a minimum with the convenient appliances now used in the most improved plants. I have had no practical experience with calcination, and therefore can express no positive opinion based upon a comparison of the two methods; but certainly the acid method, if properly conducted, leaves very little to be desired. It has been variously objected to as unhandy and inconvenient; the chief objection seems to be the losses of gold and difficulties in melting, due to the presence of

zinc sulphate. The latter objection may be eliminated wherever there are good facilities for thoroughly washing the zinc slimes. Where the proper facilities do not exist, nothing more disagreeable and unhandy can be conceived than acid treatment.

One authority writes:* "The separation of the zinc by solution in sulphuric or hydrochloric acid, or in acid sodium sulphate, has been stated to be impracticable owing to the difficulty of filtering the slimes. Moreover, the vessels in which the solution is effected boil over from the copious evolution of hydrocyanic acid, and there are other drawbacks to this method."

The difficulties with acid treatment may be assumed to be chiefly mechanical; and have, within the last year or two, been so far overcome as to make the method a very convenient and simple one.

The following method of treating precipitates in South Africa is that described by the African Gold Recovery Co.:†

"The method most in use for refining gold slimes in the South African gold fields is by the use of nitre. The slimes are dried till just before they become dusty; they are then mixed with powdered nitre, the amount varying from 3 to 33% of their weight, and gently heated as a thin layer, either in a wrought-iron pipe (10 in. in diameter by 6 ft. in length), or preferably in a tray of wrought iron (⅜ in. thick, by 6 ft. by 3 ft. by 1 ft.), which may also be used for the drying process. In neither case do the flames come into direct contact with the slimes; a hood carries off the obnoxious fumes. By the use of nitre everything in the zinc precipitation boxes which passes a sieve of three or four meshes to the lineal inch may be refined, and thus the finely divided zinc, which otherwise accumulates and clogs in the boxes, is constantly removed. Less nitre is always used than is required to oxidize all the base metals present, as otherwise the free nitre will rapidly corrode the plumbago crucibles, which subsequently are used for melting; it is advisable, however, to remain as near as possible below the limit, as the roasting which follows is thereby conducted quicker and at a lower heat. Besides rendering the bullion finer—containing say only 15% base metals—this nitre-roasting gives a cleaner slag, and lessens by at least one-half the time required for fusing the gold slimes, and prevents violent ebullitions of vapor from the crucible. From 3,000 to 4,000 oz. of bullion can be obtained in 24 hours from roasted slimes containing 33% of gold by the use of No. 70

* Rose, "Metallur y of Gold," p. 305.

* Dr. S.heidel's Monograph on the Proce s, p. 54.

plumbago crucibles, with good coke, in four-box furnaces (20 in. square by 22 in. deep). The following fluxes have been found to answer well: When much metallic oxide is present—slimes six parts, borax four parts, soda two parts, sand one part. When little metallic oxide is present—slimes three parts, borax one part, soda two parts, sand one part. The function of the sand is to form a fusible slag with the soda, and also to protect the pots against metallic oxides and the potash formed by the reduction of the nitre. The slag resulting from melting slimes usually contains an appreciable quantity of gold. This, in the absence of smelting works, is generally crushed by hand in a mortar or by power in a smallest size Gates or Fraser & Chalmers sample grinder. It is then panned, and the tailings resulting, still rich, as a rule, are shipped to Swansea. In estimating the cost of a flux, it should be remembered that a very small percentage of gold in the slag will pay for an improved flux, and that flux which gives the cleanest, most fluid, slag is preferable."

A System of Cleaning Up.—The system which I am about to describe is, with few modifications, the same as adopted at the Standard Consolidated Mining Co.'s Works at Bodie, where its success has been marked. Cleaning-up facilities were not included in the original design of the Standard plant; and the arrangement of the zinc-boxes is such as to render it impracticable to draw off the slimes in the proper way—through side launders into the acid tank. In lieu of this, the contents are removed by buckets filled from a center discharge at the bottom of each compartment—a system which works fairly well, and which, under existing circumstances, it would hardly pay to alter.

Certain important features in the system, as at present practiced in Bodie, were suggested by T. H. Leggett, formerly manager of the Standard Works, who has since developed them with success, at the Treasury Mine on the Rand.* In Bodie practice, however, the use of a Johnson filter-press for filtering and partially drying the slimes was found impracticable owing to the fine state of division of the precipitates, and the impermeability of the first thin layer forming on the filtering disks. In its stead, was provided a filter-box constructed on about the same principle as those used in agitation plants in New Zealand and elsewhere for filtering the ore. (Fig. 11.) This necessitated certain modifications in the details of operation which will be described in full.

* "Transactions Institution of Mining and Metallurgy," vol. v., p. 147.

Details of System: Washing the Zinc.—The discharge from the gold tanks into the zinc-boxes being shut off, a stream of water should be turned into the boxes, to displace the solution. Each compartment should be treated separately; and the zinc washing done, as far as possible, in the compartment, to avoid losses in moving the material about. Raise the screen and shake it about in the solution for several minutes until most of the loosely adherent precipitate shall have fallen to the bottom of the box. Then, with the hands protected with rubber gloves, give the zinc a more thorough washing, agitating it in the water, but not too roughly, or the brittle shavings will become unnecessarily broken up. It is difficult to tell just how long to continue this preliminary washing, as the water will at once become too black to show how far the shavings have been cleaned; but in most cases about five minutes will suffice. The plug at the side is then carefully and gradually drawn (to prevent a violent gush into the launder), and the accumulated slimes are carried down into the acid tank. The plug is then replaced. A stream of water from a ½-in. hose is turned in on top of the partially cleaned zinc, and the compartment filled up again. The zinc is again rinsed and rubbed in this second wash, which is in turn drained off. Three such washes will usually be enough to cleanse the box pretty thoroughly of slimes and short zinc.

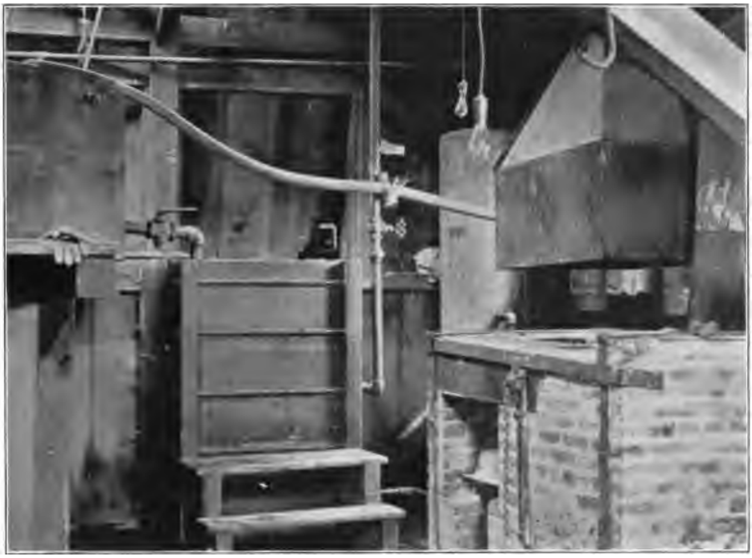
If we wish only to treat the finest slimes, a 40-mesh screen may be placed at the end of the launder discharging into the acid tank; and the fragments of zinc thus retained may be replaced in the boxes. But we have already explained the objection to this system. Alfred James* tells us that "with a 40-mesh screen and well-turned zinc, bullion has been obtained over 960 fine, without any acid treatment, roasting, or special fluxes." He was probably alluding to some very unusual case not met with in common practice. In Bodie, screens as fine as 50-mesh were tried; but there was always a certain percentage of zinc found in the slimes, which required a considerable acid treatment to eliminate.

Each compartment being washed out as explained above, and the launder finally cleaned with a jet from the hose, zinc from the end compartments is then distributed at the head, and the rest of the box replenished with fresh shavings.

It will require about as much water to wash a zinc-box thoroughly (size suitable for 75-ton plant) as can be accommodated in a 400-

* "Transactions Institution of Mini g a d Metallurgy," vol. iii., p. 404.

PLATE VIII.—CLEAN-UP AND WASHING ROOMS STANDARD WORKS, BODIE, CAL.



CLEAN-UP ROOM AT PLANT NO. 1. SHOWING ACID TANK, FILTER-BOX, WATER-HEATER, OPEN FURNACE AND HOOD.

CLEAN-UP AND WASHING ROOM AT PLANT NO. 2, SHOWING GASOLINE ENGINE, VACUUM PUMP, FILTER-PRESS AND FILTER-BOX. (See p. 62.)

gal. acid tank. Therefore, as soon as a box is washed, and while fresh zinc is being distributed, the liquid in the tank may be allowed to settle, and in about half an hour the supernatant liquor, practically clear, may be decanted off into the settling tank with a 1½-in. 5-ply hose. The second zinc-box may then be treated in the same way.

Where the clean-up takes place only once a month, and is unusually large, a more capacious acid tank may be required; or the clean-up of the weak solution boxes may take place at another time. The bulk of precipitate will be found to vary in proportion to the amount of silver present. These are matters which must be determined by practice.

When all the boxes are cleaned, the precipitate is allowed to settle in the acid tank, and the clear liquid siphoned off into the settling tank. In the meantime water is being heated in the boiler; and when the contents of the tank have been decanted down as close to the slimes as possiblé, the material is ready for acid treatment.

Method of Applying Acid.—In some mills the acid treatment is carried on in iron vessels; so arranged that heat can be applied underneath. This is superfluous, however, as sufficient heat for practical purposes is generated by the reaction of the acid and zinc, with the addition of hot water. The method of using the acid varies. Sometimes it is diluted before being thrown into the tank. Theoretically the most efficient dilution would appear to be about 25% commercial concentrated acid; but when such a dilution is used on a large scale, the action is too slow and tedious to be practicable.

The method employed in Bodie practice is as follows: A sliding door in the roof directly above the tank is opened, as well as a door near the opposite end of the clean-up room, to insure a good draft to carry away the fumes. A conical Russia-iron hood, arranged so that it could be raised or lowered, was tried and discarded; it did not appear to furnish a wide enough channel to carry away the vast amount of vapor evolved by the first contact of acid with zinc. When everything is ready, a bucketful of acid (weighing about 30 lb.) is thrown into the mass of wet slimes. The first effervescence is usually very violent, especially if the cyanogen residues have not been properly displaced from the boxes before cleaning up; but if the draft is good, the fumes rise in a straight column, and the operator may stand alongside the tank without inconvenience.

The slimes are from time to time stirred up with a long wooden hoe until the action of the first charge of acid somewhat subsides. From now on, only half a bucketful need be used at a time; and with the addition of every fresh charge about the same quantity. in bulk, of hot water may be run in from the faucet over the tank, This process is continued, about 15 lb. at a time, with occasional stirring, until further addition of acid produces only a mild effervescence. The mixture may then be left undisturbed for about two hours. A quiet effervescence will continue until all the small particles of zinc are consumed. The end of the reaction may be determined by adding a little acid. If no effervescence takes place it may be taken for granted that virtually all the zinc has been consumed. At Bodie, where from 1,500 to 3,000 oz. of precipitate are refined at each clean-up, the actual consumption of zinc by acid requires from four to six hours. The time required will obviously depend upon the quantity of zinc present, the shape of the acid tank, the quality of the acid, etc.

Elimination of Zinc Sulphate.—When the effervescence has subsided, the black, thick mixture in the tank is diluted to within a few inches of the top, with hot water—a slow stirring being kept up while the tank is filling. The object of this second step in the operation is to wash out the zinc sulphate residues which occur as a product of the acid reduction of the zinc. If allowed to remain, they very materially increase the difficulties and cost of melting, and may cause some loss of gold from volatilization. Consequently, upon the thoroughness with which the washing is done depends much of the success of this mode of treatment.

The tank thus filled is allowed to stand from four to six hours, until the supernatant liquor may be siphoned off almost clear into the settling tank.

A second wash of hot water is then applied, and a series of such dilutions kept up, until the sulphate of zinc is practically eliminated. From five to eight dilutions will usually be necessary.

The following table indicates the value per ton left in the first hot water dilution after successive stages of settling:

	Gold.	Silver.	Total.
First hour	$0.41	$0.04	$0.45
Second hour	.20	.04	.24
Third hour	Trace.	Trace.	....

When the final dilution is drawn off down to within a few inches of the precipitate, the residues may be discharged into the filter-box. The wooden hoe may be used to push the material along the inclined bottom of the tank to the discharge valve. Any adhering material may be carried down by a jet from a small hose, and the tank thoroughly cleaned out. This proceeding will have to be carried on by degrees, as it is not expedient to fill the filter-box to a depth of more than 5 or 6 in. at a time.

Filtration.—After the steam or air-suction is applied, filtration may be hastened by occasionally scraping the blanket with a smooth-edged Russia-iron scoop. It will generally be found safer as well as more convenient to use two thicknesses of blanket as a filter. When the slimes are filtered it is a good practice to spread the mass out evenly over the filtering surface and cover it to a depth of several inches with steaming hot water, which may be run through before the final transfer to the drying pan. When the slimes are filtered, they appear as grayish-black lumps, and contain from 20 to 40% of water.

With coarse precipitate, washing and filtration may be carried on much more rapidly by means of a Johnson filter-press, into which the various dilutions of hot water may be directly pumped, without waiting for them to settle. In Bodie practice, this apparatus, as already explained, could not be used for this purpose; the one introduced is now in service at the more recently constructed of the Standard Company's plants, for filtering the slimes before acid treatment.

Drying.—At the Standard Works the precipitate is dried in an iron pan over a brick furnace, the apparatus being similar to that described on page 65. The material is dried by a slow heat. The fumes of mercury, sulphur, etc., are carried off by means of a rectangular, Russia-iron hood connected with a 5-in. stovepipe which leads to the roof. The precaution was taken of tying a thickness of fine-mesh cloth over the mouth of the pipe to catch any escaping dust. At the end of several months the pipe was taken down and cleaned, and practically nothing was found but a considerable amount of quicksilver. If the zinc be properly reduced, the sulphates removed by washing, and the slimes left undisturbed while drying, there is small probability of any loss. The drying may be hastened by cutting up the contents of the pan into small sections with a spatula. This should be done at the start, and the material not turned over or disturbed while drying.

It requires from 12 to 15 hours over a slow fire to dry from 150 to 200 lb. of precipitate. It seems probable that with some form of oven or muffle this material might be dried in less time, and with less consumption of fuel.

Cost of Acid Treatment.—The following data may be of interest:

COST OF ACID-TREATMENT OF PRECIPITATE.

Yielded by { 42,223 tons tailings—Plant No. 1. / 21,307 tons tailings—Plant No. 2. } Bodie.

Item.	Quantity Consumed per Ounce of Precipitate Recovered.		Cost in Cents per Ounce of Precipitate Recovered.		Cost in Cents per Ounce of Gold Recovered.		Cost in Cents per Ton of Tailings Treated.	
	Plant 1.	Plant 2.	Plant 1.	Plant 2.	Plant 1.	Plant 2.	Plant 1.	Plant 2.
Sulphuric acid...........	0.1113 lb	0.1040 lb	0.45	0.42	3.83	4.74	0.66	0.61
Wood consumed for steam vacuum-ejector and for drying precipitate....................	0.0006cds	1.0005cds	.45	.37	3.83	4.18	.66	.54
Labor.....................	....	....	.28	.24	2.38	2.71	.41	.35
Total................	....	....	1.18	1.03	10.04	11.63	1.73	1.50

Practice.—The precipitate pan and contents should be cooled, and moved directly to the melting-room without transferring the material to other vessels. Losses from dusting are due largely to unnecessary handling of the finely divided, extremely light precipitate. In some works the precaution is taken of reducing the whole mass to a powder, in the belief that this simplifies subsequent operations. A plan which has been found to work well in practice is as follows: The dry precipitate is scored, and cross-scored with a knife to cut up the larger lumps. A certain quantity (100 oz.) at a time is carefully transferred into a pan and weighed, and the necessary flux added. From the pan the material is cautiously shoveled into No. 60 plumbago crucibles, a good practice being to have two or more crucibles in service at the same time. As the pots are being charged, the flue-damper is shut off to avoid dusting. At Bodie a dust-chamber has been interposed between the furnace and the stack. This was cleaned out after about 40,000 oz. of precipitate had been melted, and the dust recovered showed a value of only $50. The losses from dusting, where the proper precautions are taken, may be set down as practically *nil.*

Fluxing.—Owing to the complex character of the zinc-precipi-

tate, a suitable flux can only be determined after a good deal of actual experiment. At Standard Plant No. 1, where the cyaniding of tailings is uncomplicated, and the zinc-precipitate contains 36% silver and 11½% gold (average) the following flux is used: With every 100 oz. precipitate is mixed 50 oz. flux, containing 4 parts borax, 2 parts soda, and 1 part sand.

The tailings treated at Plant No. 2 contain complex organic compounds which foul the solutions and slowly precipitate out in the gold tanks and zinc-boxes. Some of this material unavoidably gets mixed up with the gold slimes, and when present in any great quantity adds to the difficulties and the time consumed in melting. The precipitate from this plant contains 36% silver and 9½% gold (average). With every 100 oz. is mixed 75 oz. of the same flux used at No. 1 Plant.

The tailings treated at the South End Cyanide Co.'s Works, Bodie, contain a large proportion of silver; and for the very bulky precipitate taken from the zinc-boxes the following flux has been found serviceable: ⅜ lb. borax, ⅛ lb. pearl ash, and $\frac{1}{20}$ lb. of 2 parts soda and 1 part carbon, to every pound of precipitate.

Details of Slag and Bullion Yield.—The following tables give details of slag and bullion yield at the Standard Company's Plants during the whole period of operations:

SLAG. (*a*)

Standard Plant.	Average Value Slag per Pound.			Pounds of Slag per Ounce of Precipitate.	Pounds of Slag per Ounce of Bullion.	Value Slag (in cents) per Oz. of Precipitate.	Value Slag (in cents) per Oz. of Bullion.
	Gold.	Silver.	Total.				
No. 1........	$0.66	$0.10	$0.76	0.039	0.075	3.01	5.84
No. 2........	.40	.09	.49	.031	.057	1.55	2.85

(*a*) Refining charges at smelter about 12c. per lb.; transportation charges about 10c. per lb.

PRECIPITATE AND BULLION.

Standard Plant.	Number of Melts.	Average Fineness of Precipitate.			Average Fineness of Bullion..		
		Gold.	Silver.	Total.	Gold.	Silver.	Total.
No. 1..........	51	115.4	360.1	475.5	226.4	701.5	927.9
No. 2..........	19	95.0	364.5	459.5	187.0	717.6	904.6

COST OF FLUXING AND MELTING PRECIPITATE.

Item.	Quantity Consumed per Ounce of Precipitate.		Cost in Cents per Ounce of Precipitate.		Cost in Cents per Ounce of Gold Recovered.		Cost in Cents per Ton of Tailings Treated.	
	Plant 1.	Plant 2.	Plant 1.	Plant 2.	Plant 1.	Plant 2.	Plant 1.	Plant 2.
Charcoal	.0131 bu	.0124 bu	0.30	0.28	2.55	3.16	0.44	0.41
Borax	.285 oz.	.43 oz.	.20	.29	1.70	3.27	.29	.42
Soda	.143 oz.	.21 oz.	.25	.35	2.12	3.95	.37	.51
Sand	.07 oz.	.10 oz.	....	....	....	....	....	....
Graphite crucibles	....	....	.08	.08	.68	.68	.12	.11
Labor	....	....	.47	.40	3.99	4.52	.69	.59
Total	....	....	1.30	1.40	11.04	15.58	1.91	2.04

CHAPTER XI.

TECHNICAL RESULTS.

THE cost of labor in operating cyanide mills in the United States is a very significant item in the total cost, and has much to do with the economic possibilities of the process. The secret of the comparatively low cost of treatment in South Africa is not so much in the small consumption of chemicals as in the low cost of labor. In Bodie, which is a high-wage camp, the cost of labor amounts to between 35 and 45c. per ton of tailings treated, but decreases inversely as the tonnage, having reached in one month as low as 16½c. per ton.

It is quite impossible to specify what the working force should be in a plant, the conditions being so variable. Experience will determine in each case just what force is indispensable and what special work to apportion to each employee.

The following force is employed at each of the Standard (75-ton) Plants, Bodie: Day shift—one superintendent (for the two plants); one foreman; one engine man; one laborer (employed raising dams to retain discharged residues); one laborer (employed in recording weight of wagon-loads of tailings and distributing lime); the cartage is done by contract at so much per ton. Night shift—one foreman; one assistant.

The foreman should keep an accurate tabulated account of the progress of each vat in the cycle of operations on a blank form provided for the purpose. Such forms supply a complete detailed record of operations, and are referred to by the superintendent in making his monthly statement. The foreman should also take charge of the assay samples, determine the moisture in a charge, and make the necessary sizing test. He also makes frequent tests on the strength of solutions, and directs the solutions into their proper channels. Once a day he inspects the zinc-boxes and shifts the zinc from the lower compartments.

The engine man attends to the pump and power plant, and assists the foreman at the clean-up.

Sluicing out, wherever possible, should be done at night, so that charging vats can be carried on without interruption during the daytime.

The duty of the superintendent should be to supervise the general working of the plant, and to study how to increase its efficiency and decrease the cost of treatment. He need not be a skilled chemist, but he should be conversant with cyanide practice, and possess a general knowledge of chemistry and metallurgy, and be able to carry on laboratory tests with intelligence and discrimination.

A tabulated form for daily use in a plant should be provided with space for the following details: Number of charge, date of charge, moisture in charge, hours taken up in charging, time of introducing strong solution, time of saturation and of the various stages in the leaching process, time of discharging, time taken up in discharging, daily consumption of chemicals, and the total number of hours for complete operations in each vat.

The superintendent's statement should contain a summary of the details of operation during the month. The form used by the Standard Company, which is given on opposite page, is a very complete and practical one.

Cost of Treatment.—In the operation of the cyanide process which has hitherto found its greatest field in the treatment of low-grade tailings where the margin of profit is necessarily small, the cost of treatment is obviously a matter of first importance. A number of factors enter into it, the most important of which are the following: (1) The quantity of material treated; (2) consumption of chemicals (chiefly cyanide); (3) freightage on chemicals and supplies; (4) cost of labor; (5) cost of transporting tailings or ore; (6) cost of cleaning-up and transporting bullion; (7) cost of power and water-supply.

It is obvious that the total cost will vary in different localities; and that any generalizations as to the comparative cost of cyanidation, chlorination, and other methods of treatment are purely idle, being matters wholly dependent upon local conditions. A plant located near a railroad, where freighting facilities are good, might stand a higher consumption of chemicals than one inaccessibly situated; or the consumption of chemicals, and the freightage charges may both be prohibitively high.

Resumé of Operations at Tailings Plant No.........

For the Month of................189..

Dry Tons Treated,		
Per Cent. Moisture,		Coarse, %
Sizing Test (80-mesh screen),		Fine, %
Assay Value Charged Tailings,	Gold	
(at c. per oz.)	Silver	
	Total	
Assay Value Discharged Tailings,	Gold	
	Silver	
	Total	
Extraction Indicated by Assays,	Gold	
	Silver	
	Total	
Extraction Obtained in Product,	Gold	
	Silver	
	Total	
Percentage of Extraction Indicated,	Gold	
	Silver	
	Total	
Percentage of Extraction Obtained,	Gold	
	Silver	
	Total	
Total Product Indicated by Assays,	Gold	
	Silver	
	Total	

Cost of Treatment.

Item.	Quantity.	Value.	Quantity per Ton.	Cost per Ton (Cents.)
Cyanide, lb.				
Zinc, lb.				
Lime, lb.				
Sulphuric Acid, lb.				
Gasoline, gal				
Wood, cords				
Coal Oil, gal.				
Misc'laneous				
Tot. Supplies				

	Total Cost.	Cost per Ton, Cents.
Tot. Supplies		
Labor at Plant.		
Labor at Shops.		
Hauling.		
Assaying and Melting.		
Transpo'ting and Refining Bullion.		
Totals.		

Product.

Ounces of Precipitate.	Bar No.	Weight in Ounces	Gold	Silver	Total

Remarks:

The chief study of a cyanide operator should be to reduce the cost of treatment as low as possible. In Western America, where the application of the process is comparatively new, it is a matter of some difficulty to obtain accurate figures on cost of treatment, owing to the reluctance on the part of managers or companies to publish results which they feel can be improved upon, and the lack of accurate details obtainable from plants actually in successful operation.

Owing to this lack of published results, and the failure of many attempts to apply the process profitably, the idea has become current that the cost of treatment in the United States is too high to make the process applicable on a large scale; and that the ore is generally unsuitable for cyaniding. This is one of those sweeping generalities which cannot be taken seriously, like so many popular views pertaining to mining and metallurgical processes. The limited application of cyanidation in the West is probably due quite as much to a lack of practical knowledge of the process, and ignorance on the part of those assuming to pass upon the availability of samples of ores or tailings for cyanide treatment, as upon any real unfitness of low-grade material. In many instances absurd plants have been erected, and operations actually commenced, under the advice and direction of some plausible enthusiast, who, with a new method of precipitation, or some other vast improvement on the original process, easily imposes upon ignorant and credulous owners. Such frauds are common, however, in all professions; and would not be mentioned in this connection if it were not that the cyanide process, being so new, and so various in its mode of application, seems peculiarly infested with these so-called "experts."

In 1893 W. G. Shaw put the cost of treatment in South Africa at $1.65 per ton. Since that time the cost has been reduced, owing to a reduction in the price of cyanide, and to various economic improvements in treatment. Dr. Scheidel gives some interesting figures on cost of treatment in various mills, which have no doubt been much reduced since the publication of his work, and which, therefore, can hardly be accepted as applying to present conditions. They indicate a very variable cost; and are chiefly interesting as showing upon how many circumstances the cost of treatment actually depends. At Mercur, Utah, the treatment by cyaniding of ore averaging about $15 per ton was at one time given out as costing $2.40 per ton. At the present time it is considerably less.

Some instructive figures have been published on the cost of treatment at Harquahala, Arizona, in the cyaniding of 42,730 tons of tailings.* They indicate a total cost of $1.40 per ton. The total yield per ton being $3.21, a balance of profit was thus left of $1.80.

At Bodie the cost of treatment has been fairly low, considering the isolation of the camp, the high cost of freightage on chemicals and other supplies, and the high price of labor. Accurate data of cost of treatment in certain plants has been obtainable; in others, such information was inaccessible, or so imperfectly kept as to be unreliable. The cost of chemicals alone has been calculated from accurate data of the cyaniding of 73,976 tons of tailings. The other items, such as labor, hauling, etc., which vary considerably from month to month in the different plants, are given approximately:

	Pounds Consumed per Ton.	Cost per Pound.	Cost per Ton of Tailings.
Cyanide	0.46	$0.43 (*a*)	$0.197
Zinc	0.26	.13	.038
Lime	4.83	.01½	.072
Sulphuric acid	0.17	.04	.006
Gasoline	....	....	.010
Electric power	....	....	.002
Total cost of chemicals and power, per ton of tailings	....	....	$0.325
Labor (*b*)			$0.35 to $0.45
Preparing and hauling tailings			.25 to .35
Assaying and melting			.02 to .03
Supplies (miscellaneous)			.01 to .03
Refining and transporting bullion			.04 to .06
Total cost of treatment (*c*)			$0.995 to $1.245

(*a*) The price of cyanide (laid down in Bodie) has fluctuated between 58c. (1894) and 31c. (1897); 43c. represents the average price per pound.

(*b*) The cost of labor has in certain cases, owing to slimy character of material and low tonnage, been considerably higher than the figures given.

(*c*) Only the expenses connected with the actual operation of the plant are given here. The cost of treatment should properly include the patent royalty paid to the company controlling the MacArthur-Forrest patents, the interest on the original investment, and the various sums of money chargeable to the sinking fund. The royalty is extremely variable, depending upon the quantity of material treated, its value, etc.

The minimum cost of treatment in Bodie (so far as I am informed from accurate data) is that recorded for Standard Plant No. 1 during the month of October, 1895, when 4,290 tons of tailings were treated in four 75-ton vats, or 138¼ tons per day. The tailings were from an exceptionally free-milling class of ore, were easily leached, and gave a good extraction in a short time. The

**Engineering and Mining Journal*, Jan. 16, 1897 (quoted from Harquahala Gold Mining Co.'s annual report, 1896).

figures are interesting as showing that an increased tonnage very materially affects the cost of treatment; that it is attended by a decrease in the cost of labor, and that the consumption of chemicals per ton does not appear to increase in the same proportion as the tonnage.

	Pounds Consumed per Ton.	Cost (in cents) per Ton.
Cyanide	.2976	16.45
Zinc	.1839	2.39
Lime	2.7198	4.07
Sulphuric acid	.1561	0.65
Wood (for power purposes before installation of electric motor)	.008 (cords)	2.25
Labor	....	16.41
Preparing and hauling tailings	....	41.66
Assaying and melting	....	1.50
Miscellaneous supplies	....	0.49
Refining and transporting bullion	....	2.00
Total cost	....	87.87

To summarize, the cost of treatment in Bodie may be said to have varied between 87c. and $1.50 per ton in the cyaniding (up to the present time) of a grand total of something over 125,000 tons of tailings.

As I have already explained, there is no uniformity in the character of the tailings, even in the same reservoir, from 25 to 40% slimes being produced from most of the ores milled in the camp. Consequently the tonnage in the same plant may vary between 1,000 and 4,000 tons per month, according to the fineness or coarseness of accessible material—a slimy charge requiring sometimes as long as 10 days of contact with slowly percolating solutions.

It would be a mistake, however, to cite Bodie practice as an instance of successful application of the process, without pointing out the fundamental reason for this success, namely, the extremely favorable conditions existing in the material itself. Not even excepting South Africa, there is probably no locality where the simple cyaniding of tailings is carried on with such satisfactory results; there being nothing in the ore to complicate the process beyond a small percentage of decomposition products. The extraction of the gold is between 75 and 80%, and the precipitation on zinc shavings almost perfect. It would simplify the process very much if all material yielded its gold so readily; unluckily, however, the application of the process in many localities has been

fraught with immense difficulties, and chemical and metallurgical problems have arisen which have baffled experienced chemists and operators.

Extraction.—The percentage of gold extracted by the process is extremely variable, depending on conditions alluded to in previous pages. So far as can be ascertained from official reports it appears to be about 70% of the assay value of tailings; in the direct treatment of ores much higher results are claimed, in many instances over 80%. A careful study of working details has undoubtedly resulted in an increase of extraction during the past few years. It is, however, by no means satisfactory; it seems probable that with a more intelligent study of the chemical and mechanical features of the process very much more perfect results will in time be obtained. W. G. Shaw, writing in 1893, says: "After a careful course of inquiries (in South Africa) I consider the average cost of treatment by cyanide to be $1.65 per ton, with an average extraction of 67% of the gold." Feldtmann,* writing a year later, says. "The average extraction from acid ore, say from 5 to 6 dwt. stuff, may be put down at 70%."

Concerning the operations at Mercur, Utah, Louis Janin, Jr., wrote, some time ago: "The extraction at the Mercur has varied. When the mill was first put in operation it was considerably below 70%, but as experience with the process increased the results became more encouraging, until now, I am informed, the average return is between 85 and 90%. The mill has been enlarged, and better results are anticipated, both as to extraction and cost."†

The official report of the Harquahala Gold Mining Co. (Arizona) for 1896 shows a saving of 77% of the assay value of the tailings treated.

In Bodie the extraction of gold varies between 70 and 80% of the assay value of the tailings; and of silver, between 30 and 40%.

Dr. Scheidel is authority for the statement that in New Zealand the results of cyaniding ores and tailings are in general better than in South Africa. This statement, however, was made in 1894, and may not hold good for later practice.

Information concerning operations in New Zealand is given by Dr. Scheidel as follows:‡

"The Waihi ores, pure quartz, the gold free, but exceedingly fine, the silver in form of sulphides, no sulphurets of base metals,

* *Engineering and Mining Journal*, August 4, 1894. † *Ibid.*, Oct. 7, 1893.

‡ "Bulletin No. 5, California State Mining Bureau," p. 41.

give an extraction of from 85 to 91% of the gold assay value, the silver returns varying from 43 to 51%. The ore of the Crown mines, which resembles those of Waihi, but containing occasionally telluride of gold, yields on an average 93% of gold and 79% of silver. Concentrates, if satisfactory at all in cyanide treatment, give as a rule very high figures. A considerable quantity of concentrates from the Sylvia Mine, in New Zealand, of a very complex character, being composed chiefly of zinc-blende and copper pyrites, with a large percentage of galena and iron pyrites, were treated by me by cyanide, and gave very satisfactory results under conditions where no other means of treatment were at disposal. The said concentrates are classified by the dressing plant; the fine slimes, rich in bullion and galena, gave as high an extraction as 95.43% of the gold and 86.69% of the silver; coarse concentrates gave an average of 80.32% of the gold and 50% of the silver. A large parcel of very fine sulphurets (from the canvas plant) from the Utica Mine, California, consisting of pure iron pyrites in finest division, mixed with more or less fine sand and carbonate of lime, proved an excellent material for cyanide treatment; the extraction averaged 93.18% of the gold value, rising in some instances as high as 96.57%. The coarse concentrates from the Frue vanners did not give such good results, if treated direct; their reduction to greater fineness, however, improved results."

Gold Losses.—Gold losses in a cyanide mill are traceable to a number of sources, which, for purposes of convenience, may be divided into real and apparent. Among real losses are the following: (1) Leakage from vats, zinc-boxes, launders and pipes, overflows; (2) absorption of gold by wooden vats; (3) losses during cleaning up; (4) loss by dusting during drying of precipitate; (5) loss by dusting and volatilization during melting; (6) passage of particles of precipitate into general circulation, from careless manipulation of zinc-boxes.

(1) In a carefully constructed and operated mill there should be no appreciable loss from leakage. Small leakages from vats and launders and overflows from gold and storage-tanks are apt to occur; their importance will depend upon the value of the solution. Ordinarily the working solution in a tailings mill carries only from 4 dwt. to 10 dwt. of gold per ton (240 gal.) when leaving the leaching vats; consequently the leakage or overflow would have to be very large to mean much of a loss in gold.

Leakages from zinc-boxes may be more serious, owing to the presence of particles of precipitate in the solution.

(2) Losses due to absorption have probably been much exaggerated. Such losses may occur in two ways: (*a*) By the mere retention in the wood of a certain amount of solution, varying between 20 and 40% of the weight of the wood; (*b*) by a series of absorptions, followed by evaporations, leaving behind each time a residue of gold—a state of affairs which may be assumed to obtain in leaching vats, which are drying while being charged, and in solution tanks, in which the height of the liquid is constantly changing.

In the first case it is obvious that even if a large leaching vat retains in its staves and bottom so much as two tons of solution, it cannot absorb any serious amount of gold, unless the solutions be extremely rich.

It would appear that the loss might be considerable if we assume the second theory to be the correct one. A series of tests carried out by the writer on the basis of these two theories of absorption showed that the second was hardly better than the first in accounting for any considerable loss.

Absorption may to a great extent be prevented by giving the vats a thorough coat of some form of preservative.

(3) Losses during cleaning up were probably more noticeable during the experimental days of cyaniding than they are at the present time; although they may still be considerable in poorly equipped mills, where the operation is carried on by careless or inexperienced workmen. The loss will usually be in proportion to the amount of handling; a clean-up, however, as already noted, may be carried on with only a minimum of handling.

(4) Toward the end of drying, a light, impalpable dust is apt to be carried off, if there is any attempt made (as there generally is, if an open furnace is used) to hasten the process by shoveling over, or otherwise disturbing the material. This is hardly necessary, if the drying is done in a muffle—where it can be accomplished more uniformly, much faster, with less consumption of fuel, and with less tendency to loss from dusting or volatilization.

(5) If the precipitates have been imperfectly refined, and a considerable percentage of zinc remains, the loss by volatilization may be appreciable; but it cannot be very serious where the melting is carried on in a closed pot. In charging the pot some dust may be carried up the flue and lost. This may be prevented by the use of a damper and dust chamber.

(6) If the zinc shavings are disturbed while a stream is passing through the box, or carelessly handled in being moved from one compartment to another, there is very apt to be some escape of precipitate into the sump. This material passes into the body of the solution, and is not recovered. Such losses may be obviated by taking the necessary precautions, and by the use of a screen in the last compartment of a box.

It is safe to assume that the greatest losses or discrepancies are more apparent than real; that is, due to mistakes in sampling, assaying, weighing, etc., and not to defects in the actual working of the process itself.

Such errors may be set down as follows: (1) Imperfect weighing of the vat charge; (2) inaccurate estimation of moisture; (3) the extreme difficulty of procuring an average sample from a large charge or discharge; (4) inaccurate assaying; (5) imperfect precipitation; (6) omitting to allow for the residue of value in a zinc-box after the clean-up.

I have already referred to the extreme care necessary in sampling and weighing. The assaying may be responsible for certain discrepancies, especially in estimating the charge—and discharge—value of low-grade material.

Imperfect precipitation may result in clean-up discrepancies, by allowing too great a value to pass into the working solutions. This source of gold loss, however, is not apt to be overlooked in a well-conducted mill.

The value remaining in the zinc-boxes after a clean-up is likely to be ignored in striking the balance between the extraction indicated by assays and that actually obtained. This value may be considerable, and can only be estimated by destroying all the zinc in the boxes—a system pursued in localities where work is suspended during winter, and where a "final clean-up" is made at the end of a season's campaign.

Dangers in Working the Process.—Notwithstanding the fact that cyanide of potassium is a deadly poison, the recorded mortality from accidents in cyanide mills is remarkably small. This is due to the extremely dilute condition of the solutions generally used. If a workman takes a drink of cyanide solution by mistake, the disagreeable taste generally warns him in time; and he suffers nothing more, in consequence, than a rather violent fit of vomiting and great subsequent weakness. There are several antidotes for cyanide poisoning. Freshly precipitated carbonate of iron is rec-

ommended. This is obtained by mixing equal quantities of sodium carbonate and ferrous-sulphate—chemicals which should be kept on hand in a mill, so that the antidote may be readily made. Nitrate of cobalt has been recently advocated, also peroxide of hydrogen.

In poisoning from Prussic (hydrocyanic) acid gas, the inhalation of ammonia or ether is recommended. Such cases, however, are rare.

I have seen faintness, dizziness, and a violent headache follow the inhalation of the mixed fumes arising from contact of acid with zinc-slimes in the reduction process, where a considerable amount of hydrocyanic acid is evolved. Proper facilities for ventilation during the clean-up are imperative.

Cyanide solutions coming in contact with the skin sometimes produce a painful eruption of small boils—a trouble to which some workmen are peculiarly susceptible. This may be prevented in most cases by the use of rubber gauntlet gloves (the heavy glove, No. 14, is a good average size) whenever it is necessary to put the hands in solution. Some workmen, however, cannot even wear gloves which have a trace of cyanide on their reverse side without the appearance of the skin-eruption. This is frequently a serious matter, incapacitating the sufferer from work for several days. A good preparation for use in such cases is the compound tincture of benzoin (Friar's Balsam) applied freely several times a day.

With domestic animals cyanide poisoning in any form is almost immediately fatal. In Bodie a number of cows and horses have been poisoned by drinking creek water impregnated with weak waste cyanide solutions—and damages have been, in most instances, allowed.

It may be worth while to consider, in all cases, the destination of these waste solutions. Cows seem wonderfully susceptible to the poison, instances having been observed repeatedly of their dying in a few minutes after drinking liquids containing only a trace of cyanide. This is probably because a thirsty cow will usually drink her fill, regardless of what she is drinking; while the dog and horse are instinctively more discriminating.

CHAPTER XII.

MODIFICATIONS OF THE SIMPLE CYANIDE PROCESS.

The Siemens-Halske Process.—The objections to zinc precipitation, the difficulties of cleaning up, the impossibility, in some instances, of precipitating the gold, and the impurity of the bullion produced, led to a series of researches, to discover some better mode of precipitation. What is known as the Siemens-Halske process is one result of these investigations. To Dr. Siemens, of Berlin, is due the credit of having made known the possibility of depositing gold by electrolysis, a discovery which has been put to successful practical test in large cyaniding works in South Africa.

This method depends upon the dissolving action of an electric current on an auro-cyanide solution—liberating the metalloid at the positive pole, and depositing the gold at the negative pole. After considerable experimentation, lead was found to be the most suitable metal for the cathode or negative element, and iron for the anode or positive element. It was found, further, that the best results were obtained by giving a large surface to the electrodes, and by keeping up a slow artificial circulation of the solution.

When the weak electric current is turned on Prussian blue is generated at the positive pole (from the reaction of oxide of iron and ferrocyanide) and the gold is deposited on the lead sheets as a hard, bright, yellow film. Von Garnet, one of the principal developers of this process in South Africa, gives the following facts concerning its application at the Worcester Works:* "The precipitation plant consists of four boxes, 20 by 8 by 4 ft. Copper wires are fixed along the top of the sides of the boxes, and convey the current from the dynamos to the electrodes. The anodes are iron plates, 7 ft. long, 3 ft. wide, and 1/8 in. thick. They stand on wooden strips placed on the bottom of the box, and are kept in

*Transcript from "Transactions of Chemical and Metallurgical Society of South Africa," in *Engineering and Mining Journal*, Nov. 3, 1894.

vertical position by wooden strips fixed to its sides. In order to effect circulation in solutions passing through the box, some of the iron sheets rest right down on the bottom, while others are raised about 1 in. above the level of the solution, thus forming a series of compartments similar to those of a zinc-box, the difference being that the solution passes alternately up and down through successive compartments.

"The sheets are covered with canvas to prevent short circuit. The lead sheets are stretched between two iron wires, fixed in a light wooden frame, which is then suspended between the iron plates. The boxes are kept locked, being opened once a month for the purpose of the clean-up, which is carried out in the following manner: The frames carrying the lead cathodes are taken out one at a time. The lead is removed, and replaced by a fresh sheet, and the frame returned to the box, the whole operation taking but a few minutes for each frame. By this means the ordinary working is not interrupted at all, and the cleaning out of the boxes, which is necessary in the zinc process, is only required at very long intervals. The lead, which contains from 2 to 12% of gold, is then melted into bars and cupelled. The consumption of lead is 750 lb. per month, equal to 1½d. per ton of tailings. The working expenses, including filling and discharging tanks, come to 3s. per ton.

"In order to precipitate the gold from cyanide solutions only a very weak current is required, that is to say, a density of about 0.06 ampere per square foot. With cathodes 1½ in. apart, 7 volts is sufficient."

The following data concerning the Siemens-Halske process in South Africa has recently been published:*

"The Siemens process has continued to be introduced during the year. The slimes plants now erected are using this process and also several important tailing plants.

"Continued study in the construction of the Siemens boxes has been carried on, and now standard sizes of boxes are constructed. A plate 26 in. wide, 48 in. deep, and $\frac{3}{16}$ in. thick is used, and for sand plants the boxes are 5 ft. square and 32 ft. long, and contain 156 iron anodes, while in the slimes plants the boxes are 10 ft. wide and 5 ft. deep, and contain 24 iron anodes in each compartment, set 4½ in. apart, being 288 anodes in 12 compartments, and a current density of about 0.04 ampere is used in the slimes plants, and

*Engineering and Mining Journal, Dec. 4, 1897.

about 0.6 in the sand plants per square foot. Both lead sheets cut into strips and lead shavings are used in various plants as cathodes.

"There are now 13 plants using the Siemens process, while 17 new plants are in course of erection."

The chief advantages claimed for this process are that it enables the gold to precipitate independently of the amount of cyanide in solution; that it does away with certain zinc-box complications, such as the formation of alumina, hydrate of iron, etc., and that it is simpler, cleaner, and gives a higher grade bullion than the zinc process. In 1894 it was introduced in a number of large plants on the Rand. Excellent results were at once claimed for it, and it seemed likely, at that time, to supplant zinc precipitation altogether. Charles Butters has applied it successfully to the precipitation of gold from the extremely weak cyanide liquors resulting from his method of slimes treatment operated at the Robinson Mine and elsewhere. Very little has been published on the working results of the process, and what meager reports have come to us have been, in the main, contradictory. It is now nearly four years since Von Gernet's statement was published. During this time the development of the process seems to have steadily progressed, to the extent of making it a permanent and economic success. Its applicability, however, appears to be limited to those cases where very dilute cyanide solutions may be economically used, but where the precipitation of such solution on zinc shavings is not practicable. In other words, it furnishes a successful means of precipitating from very weak solutions.

A point taken by Von Gernet in support of this process, and one which has often been quoted, is that "a solution containing 0.03% of cyanide will dissolve gold just as effectively as a solution containing 3%, provided a longer time is allowed for treatment." This is undoubtedly true; but as 0.03%, and 3% solutions are rarely, if ever, used as extracting solutions, the argument does not seem to have much practical bearing. In using zinc precipitation, the proper strength for a strong solution is usually determined by its extracting power, and not by its availability for precipitation on zinc. If a solution containing 0.2% cyanide is found to give the best economic results in a given time, and precipitation from dilute solutions (say, as weak as 0.02%) on zinc is perfectly satisfactory, then, in this particular case, there would be no gain in efficiency by using the Siemens process.

In regard to precipitating on zinc from weak solutions, Alfred

James says:* " . . . it is proved that on the Rand and elsewhere, for years past extremely weak solutions, containing only traces of cyanide, have been regularly treated by zinc. During investigations extending over some weeks, in which samples were taken from each vat every hour and assayed, it was discovered that at the point where the moisture contained in the tailings is followed by the cyanide solution, the latter frequently contains only traces, or 0.005% cyanide, though gold is present to the amount of 4 to 10 dwt. . . . I may mention that on another gold field where the matter has been more thoroughly investigated than on the Rand, owing to the employment of water for discharging the tailings, there are three systems of zinc-boxes instead of two; and the solutions passing through the third system contain only some traces to 0.07% cyanide, yet the extraction is most complete. The idea was to treat solutions which were hitherto allowed to run to waste, and the result has been a substantial addition to the returns, and the effluent seldom, if ever, shows more than 2 gr. of gold per ton.†"

In the matter of cost of treatment, we have to contend with the cost of iron and lead sheets, as against the cost of zinc shavings. We are told that in a plant treating 3,000 tons of tailings per month, the consumption of iron would be about 1,080 lb., and of lead 780 lb. The same number of tons treated by the zinc method would require about 900 lb. of zinc shavings. In cases where zinc will not precipitate from weak solutions, there is undoubtedly a considerable saving of cyanide effected in the Siemens process; only, however, where dilute solutions are found as effective as stronger solutions for leaching purposes.

A considerable saving in cost of treatment is stoutly claimed for the Siemens process; published results, however, are too meager, and mere reports too contradictory, to be relied upon. We can only conclude that the process must have some conspicuous merits, or it certainly would not have been applied on so large a scale in South Africa. On the other hand, it must have a weakness somewhere, or it seems reasonable to suppose it must have been universally applied on the Rand, where chemists and cyanide men have abundant opportunity to observe its operation. The royalty payable to the company controlling the patents may have much to do with its limited application; whereas a royalty is no longer paid for the use of the MacArthur-Forrest process in South Africa.

* "Transactions Institution Mining and Metallurgy," vol. iii., p. 96. † *Ibid.*

The apparent simplicity of the Siemens-Halske process, its recent introduction, and the inflated reports that have come to us of its success, are very apt to lead the inexperienced and enthusiastic into the mistake that electrolytic precipitation is the one panacea for a great many evils. The necessity may be emphasized here of a thorough study of the applicability of both methods before actual operations are commenced.

The Pelatan-Clerici Process.—This process has been recently introduced at DeLamar, Idaho, with considerable promise of success. It is a modification of the original cyanide process, the extraction of the gold and its recovery being accomplished at the same time in the same vat. The ore, crushed to suitable fineness, is agitated with a cyanide and salt solution in a vat provided with a mechanical stirrer. While this process is going on an electric current is introduced into the central shaft, and carried down to iron anode plates fixed to the lower part of the revolving arms. The cathode is a bed of quicksilver covering the copper bottom of the vat. The gold-bearing solution is at once decomposed, and the gold amalgamated. After the required treatment the residues are run off, and the agitator filled with a fresh charge. Once a month, or oftener, the amalgam is taken out and strained through bags as in an ordinary clean-up in an amalgamation mill.

D. B. Huntley describes the operation of the process at De Lamar as follows: "The ore is crushed, using as little water as possible; about one of ore to one of water (which we call 100% water) is suitable for De Lamar ore, and this makes a thick batter. This runs, and is scraped by a chain conveyor along a trough, and is emptied into one of the agitator vats. It is kept in slow motion to prevent settling by an ordinary agitator shaft and four arms (suspended from above, but not piercing the tank from below) until the agitator is filled to a mark previously established as giving the proper amount for a charge. Then the flowing pulp is turned into another agitator. When a precipitating vat is ready it is charged by opening a 4-in. valve, allowing the agitator charge, before mentioned, to flow through a trough to the precipitating vat. It is filled in a few minutes, and the chemicals are put in. For De Lamar ore they consist of 2½ to 3 lb. of KCy and 6 lb. of salt per ton. The charge is kept in this vat in motion about 20 revolutions per minute, for 11½ hours. A charge is counted as 2½ tons, making the capacity of one vat 5 tons per 24 hours. The electric current is turned on by a switch, passes down the agitator shaft of the vat

to the anode plate, and through the liquid for a few inches to the quicksilver beneath, decomposing the solution of cyanide and precipitating the gold and silver on the quicksilver, where it is immediately amalgamated. At the end of the 11½ hours a 4-in. gate valve in the side of the vat, a few inches above the bottom, is opened, and the pulp (then tailings) flows out. No extra water is used to wash out the remainder, but the discharge valve is closed, and the vat is ready for a new charge. This is repeated with all the other vats. The plant requires two men per shift. This includes the engineer, but not the battery and Huntington mill man.

"We have found it best to clean up twice per month, to prevent the mercury becoming too rich, which seems to make higher tailings. In cleaning-up two extra men are needed. After a charge is discharged, clear water is turned into the precipitating vat, the valve kept open, and the residue washed out—excepting a few buckets of sand which will hang. Then the agitator shaft is stopped by running the belt upon a loose pulley, the iron valve below the bottom opened, men get into the vat with brooms and sweep the sand, water, quicksilver, and thin amalgam to the discharge hole. It runs down an inclined trough into the clean-up pan. Usually around the outer part of the pan bottom about a kettle full of fairly hard amalgam is found similar to that which is scraped from parts of an amalgamating pan. From the clean-up pan the amalgam and quick are drawn off and strained through conical duck sacks as at any pan-mill. It is retorted and melted as usual, furnishing bullion about 950 fine in gold and silver."

I am indebted to Mr. D. B. Huntley, manager of the De Lamar Mining Co., Ltd., for the following notes upon the Pelatan-Clerici process as applied at De Lamar, Idaho: "The Pelatan process worked here at the rate of about 1,300 tons per month from June, when it commenced, to February 6th, when it shut down. The amount of bullion secured from it was $10,000 to $14,000 per month; and, in general, it may be said, it worked to a slightly better percentage than our old pan mill, with a slightly decreased cost.

"The mechanical trouble with it was the firm precipitation on the copper plate cathode on the bottom of the vat. When we cleaned up the copper plates for 10 vats, after the final shutdown, we secured over $9,000.

"After several months we found, much to our surprise—due to

some obscure forces—that the copper plate corroded or was dissolved, or at least was eaten through, and the quicksilver went into the bottom of the vat, which was usually of wood, and of course leaky. We made a cement bottom, and put pieces of copper plate cathode on top of it, which we found to do very well. I think finally when we start up we shall have cement bottoms in the vats, with depressions in the cement to catch and hold the quicksilver, and thus prevent its being very movable, and also use a larger amount of quicksilver, and we expect good results.

"We have a peculiar assortment of conditions, and after much experiment decided that we could do better under our conditions with straight vat leaching than with either pan mill or the Pelatan-Clerici process; hence the shut-down of those two departments and the introduction of cyanide vat leaching.

"We are not crushing our ore at all, except through a rock-breaker, and are depositing it in 3½ to 4 ft. deep vats having a capacity of about 25 tons each. It requires about four days to leach, with 0.25% of cyanide solution, and we are getting about the same percentage as formerly secured by pan mill or Pelatan-Clerici process.

"Our vat leaching is really in the experimental stage only, and I cannot give accurate figures at present."

Overwhelming advantages are claimed for this process by the company controlling the patents. It is claimed that it can be worked much cheaper than either chlorination or cyanidation—a fact which does not seem to be corroborated by any published working result. Details of the cost of treatment at De Lamar would be a matter of considerable interest. The cost of chemicals is set down in the company's prospectus as $1 per ton of ore treated. In the treatment of low-grade tailings by the cyanide process a cost of $1 per ton of tailings for chemicals alone might be, in many localities, prohibitively high. The cost of treatment is always a relative matter; and the cheapness of a method must depend a good deal upon its range of application. The Pelatan-Clerici process may be cheap in the treatment of $15 to $25 ores; but it might not be cheap in the treatment of tailings or slimes.

However, it is too early to express an opinion on the merits of the process, as compared with ordinary cyaniding. It seems probable that it will develop into a useful method of treating a class of ores of sufficiently good grade to allow for the cost of motive power, and the somewhat high consumption of chemicals. It would seem

also peculiarly adapted to the treatment of slimes, providing the cost of treatment is not prohibitive. The extraction and recovery of the gold being carried on in the same vat, it does away with the tedious decanting and other mechanical difficulties in the various methods suggested for slimes treatment.

The Kendall Process.—The necessity of oxygen to the dissolving action of cyanide solutions has formed the basis of a number of special processes, among them what is known as the Kendall Improved Dioxide-cyanide process. It is claimed that by the addition of a certain amount of sodium dioxide to the working cyanide solution, the necessary oxygen is artificially supplied, and the extraction hastened.

There seems to be abundant testimony of the success of sodium dioxide treatment in the laboratory, and I believe there are a few instances of its being practically applied on a large scale. Unfortunately we have no published data of successful practical results.

Other Modifications.—Various other modifications of the MacArthur-Forrest process have been invented and written about, but do not appear to have been applied to any extent in practice. Among these are: The Park-Whitaker Cyanide process, "for the treatment of cupriferous ores and concentrates." The ore is subjected to a chloridizing roasting, after which the soluble copper chlorides are removed by leaching with water. An alkaline wash is then applied, and the gold and silver extracted with a dilute solution of cyanide." The Hannay Electro-cyanide method, very similar in general principle to the Pelatan-Clerici process. The Malloy process, in which "sodium or potassium amalgam, formed electrolytically from a solution of carbonate in contact with a bath of mercury," is used for precipitation purposes. "The alkali metal combines with the cyanogen of the gold compound, forming an alkali salt of the cyanogen, while the gold is instantly amalgamated." The amalgam is then treated in the regular way. The Pielsticker process, in which the electric current is used for precipitating purposes. The Moldenhauer process, in which precipitation is carried on by means of aluminum. The latter, it is claimed, separates the gold from solution very quickly, and does not enter into combination with it. The inventor claims for his process that the amount of aluminum consumed is only one-quarter that of the zinc required to produce the same precipitation. The Johnston process, in which pulverized carbon in the form of charcoal is used as a precipitating agent instead of zinc. The Sulman-

Vautin process, in which the dissolving action of the solution is claimed to be accelerated by the addition of cyanogen bromide. "Potential" cyanogen is thus formed which rapidly attacks the gold, forming the aurous cyanide necessary for the soluble double salt of potassium and gold. The "potentiality" of the cyanogen is effected by a halogen instead of by oxygen, as in the Kendall and other processes. The Mulholland Bromo-cyanide process, in which bromine instead of cyanogen bromide is used for liberating nascent cyanogen.

Double Treatment.—The system of re-cyaniding the residues which have already been treated has recently been introduced in South Africa. This is accomplished by using a double tier of vats, so arranged that, after the first treatment, the residues may be discharged into the second tier. The employment of cheap labor appears to make this system profitable. We have no reliable information, however, on the details of practice.

A number of tests were made by the writer to determine the expediency of double treatment; and in each case a considerable amount of gold was recovered from the residues which it was impossible to extract by one treatment alone. It was found that certain tailings, from which it was possible to extract 70% of the total assay value, still contained a value of $2 per ton after the first cyaniding. From the residues, however, 50% of their value was extracted by a weak (0.07%) cyanide solution, thus making the total extraction 77% by double treatment. Whether it would pay to handle tailings a second time would of course depend upon the extraction by the first treatment; also upon the cost of rehandling, and the price of cyanide.

The additional yield of gold from the second treatment might be due to one, or all, of several causes: (1) To the attrition to which the sand is subjected during sluicing, thus exposing more of the previously occluded metal; (2) to the very complete oxygenation of the residues during sluicing and rehandling; (3) to the re-exposure of minute particles of metal which, during the first treatment, may have lain in such a position as to be partially occluded from the dissolving action of the solution.

The Treatment of Slimes.—In South Africa it is estimated that the wet crushing of ores results in the production of about 30% slimes. Various methods have been suggested for the treatment of this finely divided material, and the subject continues to be one

of the most important and interesting problems connected with the cyanide process.

It was at one time suggested that the only means of overcoming the slimes difficulty was in dry-crushing the ore. This, however, was shown to require an entire remodeling of the wet-crushing mills in use, at a tremendous sacrifice of capital. It would also mean doing away with plates and concentrators, which so easily save the coarse gold which could not be economically recovered by the cyanide process alone.

A number of slimes plants are now in operation in South Africa —the method of operation differing very little in each. At the Robinson works the slimes were first agitated with cyanide solution until the gold was sufficiently dissolved; then the mass was discharged into vats provided with filter bottoms, where it was subjected to combined suction and decantation. The slimes were thus left comparatively dry on top of the filter, but the wear and tear of filters, the repairs on suction pumps, etc., made the cost of treatment too high for low-grade material.

The method of slimes treatment now in use in South Africa consists briefly in the following steps: 1. The separation of the slimes from the coarser material by means of pointed boxes. 2. Settlement and concentration of slimes in very large vats by means of powdered caustic lime. 3. Agitation in cyanide solution by revolving stirrers and centrifugal pumps, in some instances aeration with compressed air during agitation. 4. Settling of the slimes when the gold is extracted, and the decantation of the gold-bearing solutions. 5. Precipitation of the gold by the Siemens and Halske process.

In some plants the two methods of agitation are combined; in others they are used separately. The tendency now is toward the exclusive use of centrifugal pumps. As aerators they are just as efficient and much cheaper than air compressors; and as agitators they require less power than the revolving stirrers.

The displacement of solutions by weak washes and water is complicated, in slimes treatment, by the presence of a large amount of liquid in the body of the slimes, which it is impossible to remove by decantation. Butters claims to have reduced the moisture in slimes—by settling and decantation—to only 34%. In displacing this moisture by a succession of washes, extremely dilute solutions are formed, which, in South Africa, could not be successfully precipitated on zinc, and yet contained too great an aggregate gold

value to run to waste. The Siemens process has been successfully introduced in these slimes plants for the recovery of gold from such solutions.

It is to be regretted that so little publicity has been given to the details of "slimes" treatment at the Robinson Mine on the Rand, and elsewhere.

In the United States the treatment of slimes consists of mixing the fine and coarser elements, the accumulations of this fine material not being large enough to warrant its separate treatment on a large scale.

CHAPTER XIII.

EXEMPLIFICATIONS OF PRACTICE: SOUTH AFRICA.

THE cyanide process was introduced in South Africa in 1889, four years after the discovery of the Witwatersrand "banket" or conglomerate beds, which now constitute the chief source of the gold output of the Rand. The exploitation of these immense beds, which were found to be auriferous for many miles, led at once to the erection of gold-milling plants on a scale unprecedented in the history of mining.

It is interesting to note that with few exceptions the Rand ores have yielded their gold readily to the most simple metallurgical processes. The gold for the most part exists in a metallic condition, finely divided, mainly in the matrices of the "banket" pebbles. The deeper unoxidized rock ("blue rock") contains on an average about 3% of iron pyrites, but even this ore yields a high percentage of its gold to amalgamation. The refractory constituents of the Rand ores are insignificant, and to this fact is due mainly the universal adaptability of the cyanide process.

In the early part of the year 1889 the problem of winning the residue of gold from mill tailings was practically abandoned in South Africa. Tailings had begun to accumulate to an enormous extent, and were not only regarded as a waste product, but as an incumbrance, which the companies were glad to dispose of at a discount. In the same year the Gold Recovery Syndicate obtained from the Cassel Gold Extraction Co., which controlled the MacArthur-Forrest patents, the right to work the cyanide process in South Africa. At a small experimental plant sent out from Glasgow, various tailings, concentrates, and blanketings were experimented upon with most promising results; and in 1890 the process was introduced on a commercial scale at the tailings works of the Robinson Gold Mining Co. In 1891 the process yielded 160,168 oz. fine gold; in 1894, 587,388 oz.; and in 1897, 618,000 oz.

Wet crushing is almost universally practiced on the Rand, the gold being won by (1) copper-plate amalgamation; (2) concentration (mainly with Frue vanners); (3) the cyanide process. The bulk of the pyrites and some coarse sand are separated by concentration, the remaining pulp being then divisible into the coarser material ("tailings") and the finer ("slimes"), each of which is now treated separately.

It is interesting to trace the progressive steps in the development of the process in the Transvaal. At first only the old accumulations of tailings were treated, and the mills for some time preserved the original system of plate-amalgamation and concentration, the tailings being impounded in reservoirs for cyanide treatment. The latter were hauled up to vats in cars on inclined tramways, subjected to a single treatment by cyanide solutions, and the residues disposed of, either by shoveling out through bottom and side-discharge doors, or by means of traveling cranes. The gold was universally won from the solutions by zinc precipitation, and the average extraction from the tailings ranged between 60 and 70%. The concentrates were treated by chlorination, and in a few instances by cyanidation.

Later on, as the old reservoirs became exhausted, the expediency of treating tailings direct as they ran from the mill-batteries naturally suggested itself. A series of experiments demonstrated the feasibility of two alternatives: direct filling by means of pointed boxes, which separated the coarser tailings from the slimes; and intermediate filling by means of revolving distributors, the tailings being conveyed to a second series of vats for cyanide treatment (for details see p. 88). In both methods, the slimes are allowed to overflow, and are either conserved in settling dams or treated directly.

Slimes.—The slimes were known to contain a considerable gold value, but their impermeability to cyanide solutions seemed for a time to put their treatment by cyanide out of the question, and they were simply left to accumulate, and were considered a waste-product. This loss gave rise to a prolonged discussion as to whether dry-crushing would not be better, economically considered, than the wet-crushing process then in use.

The advantages urged in favor of dry-crushing were that the value in the slimes could be saved, that sampling could be more accurately carried on with the uniform product obtained, and that the first cost of a dry-crushing plant would be less than a wet-

crushing plant of the same capacity. In illustration the successful treatment of dry-crushed ore in New Zealand was cited.

The disadvantages were the expense of drying the ore preliminary to crushing, and the impracticability of extracting the coarse gold by cyanide, which could be so easily won by amalgamation. It was shown further that in wet-crushing there was a slight advantage in cost, and that although in New Zealand ores were crushed dry to advantage, it was only because the gold was so infinitesimally fine that it could not be saved by the wet-crushing methods then in use.

Subsequently the difficulty was believed to have been overcome by the cyanide treatment of slimes by means of a system of combined agitation and decantation, the gold being recovered from the solutions by the Siemens and Halske process. This system was ingeniously evolved by Charles Butters and others, and practically applied on a large scale at the works controlled by the Rand Central Ore Reduction Co. Mr. Butters, in a recent paper read before the Chemical and Metallurgical Society of South Africa, says:

"At the last meeting of the Chamber (of Mines) the president took the opportunity of publicly stating the fact that the treatment of slimes was now an economic success. I think that such a statement from such an authority as the president of the Chamber of Mines of the South African Republic, on a question of so much economic importance to the mineral industry, marks an epoch in the metallurgical world which is well worthy of our consideration."

Convincing as this statement appears, it is hardly confirmed by the report of the Rand Central Ore Reduction Co. for 1897, according to which the treatment of some 48,000 tons of slimes at the Central and Crown Reef Works did not prove a success, and the treatment of 75,000 tons at the Robinson Works was only conducted at a slight profit. All the slimes treated, however, were of low grade, averaging from 3 to 5 dwt. per ton. A number of plants have adopted slimes treatment, with results which are given out as satisfactory. Unfortunately, there is a great scarcity of published data of working results, as we might expect with a process of such recent introduction. We must, however, accept as reliable the statements of South African metallurgists, among whom the opinion seems to prevail that the treatment of slimes is a success. If it is not yet an established economic success it is at least fair to assume that the problem will eventually be solved on the lines upon

which it is now being attacked; that is, by settling with lime, decantation, and the agitation of the pulp with cyanide solutions.

The possibility of treating slimes has done away with the precautions formerly taken in battery practice to prevent their formation. Screens of smaller mesh are now used, which give a finer product with more slimes—a product which the more readily yields its gold to cyanide.

The following account of slimes treatment at a typical works, the Bonanza, is interesting enough to quote in full:

"As the slimes leave the tailings plant . . . it receives a supply of slaked lime, fed into the launder which conveys the pulp to the spitzkasten. The spitzkasten (of 15 compartments) measures about 25 by 15 by 5 ft. The clarified discharge from the end of the spitzkasten flows back to the mill, while the product of the lower discharge runs into a circular vat with a peripheral overflow. The settling of the pulp is accomplished in about 2½ hours, where, without the lime, it would not take less than 12 hours.

"After settling, the clear, supernatant water is decanted off from the vat and finds its way back to the mill-service, and the concentrated slimes are pumped into agitators, where the KCy solution is mixed (at a strength of from 0.006 to 0.008, equal to a consumption of about 0.2 lb. KCy to the ton of solid slimes) with the charge and agitated by revolving arms. In about four hours most of the gold is dissolved, and the charge is pumped into a settling tank, whence the auriferous solution is decanted to the intermediate clarifying tanks. These are two in number, the lower one containing a sand filter, which removes every trace of turbidity. The pellucid, auriferous solution is then delivered at the precipitation house for electrolytic precipitation. The muddy deposit left behind in the settling tank is then treated with a weaker cyanide solution known as the first wash, about 600 tons of this solution are used daily at this plant, of which one-third reaches the precipitation house, and two-thirds are sent to the tailings plant, where it is used in the preliminary washing of the sands to be cyanided."*

Concentration.—A point of considerable importance in connection with the process was the advisability of doing away with close concentration, in view of the possibility of cyaniding concentrates. The older system was to chlorinate the Frue-vanner product, and to treat the mill-tailings by cyanide after separation of the slimes.

* Walter Renton Ingalls. 'Progress in the Metallurgy of Gold and Silver.' in "The Mineral Industry" vol vi, p. 344.

The more recently advocated practice is either (1) the cyanidation of all the pulp—the concentrates and coarse sand after separation by spitzluten, and the tailings after separation of the slimes—or (2) the omission altogether of concentration, and the direct treatment by cyanide of all the pulp running from the plates. The consensus of opinion among metallurgists seems to be that close concentration is a failure, especially with low-grade ores, except where the pyrites are shown to be unaffected by cyanide. The resulting tendency has been to omit concentration, except by means of spitzluten. It has been pointed out that the latter possess the great advantage of a trifling cost of operating, and of furnishing a product which may, with comparative facility, be treated by cyanide.

At the Crown Reef works, it is claimed that an extraction of 92% is obtained from pyrites and coarse sands. This success in cyaniding pyrites has very much curtailed the usefulness of chlorination works, which hereafter will probably only be resorted to in special cases.

Direct Treatment of Tailings.—We have spoken elsewhere of the system of double or intermediate treatment practiced at the best works. When this system was first introduced the intermediate settling vats and the leaching vats were erected on the same level, as at the new Simmer and Jack works (see illustration), the tailings being discharged from the bottom of the settling vats, and hauled up in trucks to the leaching vats. What is thought by many operators to be an improvement on this method has been recently introduced, namely, the erection of the two systems of vats on a double tier, one above the other. Even this system, however, appears to have its disadvantages. "It is a matter of opinion," says one writer,* "whether the arrangement of the tanks, one above the other, is economical. Personally, I prefer an arrangement which admits of discharging into trucks, and tramming these to the second treatment vats, where their contents can be distributed with greater regularity and at a trifling cost. The plant, too, is more easily handled. As the works are constructed, one tank must wait until the other is ready for discharging, and at times this is inconvenient and likely to lead to losses of gold through insufficient washing of residues."

In less favored regions than South Africa the additional cost of

* W. Bettel, in discussion of Yates' paper, "The Cyanide Process," in "Transactions of Chemical and Metallurgical Society of South Africa," vol. i., p. 302.

tramming might be a matter well worth considering. Moreover, the necessity alluded to by the writer of sometimes having to discharge a vat before the residues are sufficiently washed is one that might be overcome by building the plant large enough so that the maximum time for washing could be allowed for each cycle of operations. If there were great variation in the time required for washing each charge, this trouble might be the less easily remedied; but where distributors are used, and the amount of slimes entering the intermediate vats is carefully regulated at the pointed boxes, there ought to be a tolerable uniformity in the time required for shoveling, washing, sluicing, etc. Of course, the whole question resolves itself into two alternatives: Whether it is better to erect a double-tier plant larger than is actually demanded by the crushing capacity of the mill battery, and so allow a safe margin for each consecutive operation in the leaching process, or to build all the vats on the same level and put up with the additional cost of tramming. In America, where the price of labor is comparatively high, the double-tier plant for the treatment of tailings will unquestionably be the plant of the future.

This system of direct, double treatment, now almost universally adopted on the Rand, has the following very distinct advantages: The distributors insure a very uniform distribution of the coarse and fine sand, and such proportion of slimes as can be economically treated with the coarser material; the tailings become thoroughly oxygenated in passing from the intermediate to the leaching vats; a considerable expense is saved, as compared with the old system of hauling from beds; and, finally, it gives a higher extraction of gold.

Electrolytic Precipitation.—One of the most noteworthy innovations in the metallurgy of the process is the recent introduction of the Siemens-Halske electrolytic method of precipitation. This was installed on the Rand at a time when many operators were dissatisfied with zinc precipitation, and were eager to try some method which would insure a cleaner bullion product, an easier-handled clean-up, and a higher percentage of precipitation. All these advantages were claimed for the new process, and it at once found zealous advocates, and was introduced in many of the largest works. Under skilled hands it has proven a very beautiful mode of recovering gold from solutions, but it seems to be a disputed question whether it possesses any distinct economical advantages over the older method. It is a matter of interest that many of the

objections to zinc precipitation, which at the time gave so popular a welcome to the Siemens-Halske process, have been in great measure overcome. (See p. 107 *et seq.*) In other words, zinc precipitation has held its own, whereas the newer method has not accomplished the results which it seemed to promise, and the attitude at the present time among cyanide operators would seem to be rather reactionary in favor of zinc. Each method, however, possesses its stanch advocates. The rather acrimonious contention to which the rival claims of the two methods have recently given rise among South African metallurgists, is one of the most interesting features in the progress of cyanide metallurgy during the past year. John Yates, who appears to be the best friend of zinc-precipitation on the Rand, has pitted himself against a formidable array of operators and metallurgists, many of whom, however, appear to be directly interested in supporting the claims of the owners of the Siemens-Halske patents. In a recent paper, Mr. Yates compares the two methods as follows: (1) A larger number of vats is required in electrolytic precipitation plants, in the proportion of about 2 to 3, owing to the slower action of dilute solutions; (2) a dynamo is required, which means an additional expense for power; (3) the lead sheets have to be sent to custom works for cupellation; (4) in general practice precipitation by electricity has proven no better than with zinc, in point of amount of gold deposited; (5) the solutions found to be most efficient in South Africa are between 0.2 and 0.3% in strength; electricity possesses no advantage over zinc in dealing with such solutions; (6) very weak solutions (with which the electrolytic method shows to best advantage) are not by any means applicable to all Rand ores; hence electricity has at best only a limited range of usefulness: it is worthy of note, however, that the present treatment of slimes is made possible by the Siemens-Halske process; (7) a comparison of extraction results in several of the largest plants shows, in general, that the best results are being obtained by zinc and stronger solutions; (8) the losses in melting and cleaning up appear to be only very little less than in the MacArthur-Forrest process; (9) the new process requires the services of an electrician to attend to short-circuiting and other troubles; (10) cutting up the lead strips is a tedious business compared with cutting the zinc shavings.

A comparison of working costs of treatment in plants using both methods shows a slight difference in favor of the zinc method. The consumption of cyanide is less in the Siemens-Halske process,

but great sacrifices are made to obtain this, which counterbalance the advantages.

The following very interesting comparison of costs (in detail) between two assumed typical plants, one using zinc and the other electrolytic precipitation, is given by Yates.* For convenience I have expressed his data in terms of American money:

MACARTHUR-FORREST PROCESS, WITH ZINC PRECIPITATION:

500 tons per day plant, costs per ton—	Cents.
Filling and discharging	20.00
Cyanide, 0.7 lb., at 27c. per lb	18.90
Lime and caustic soda	3.60
White labor	10.00
Native wages and food	3.80
Fuel and power (including haulage)	6.00
Zinc	1.50
Stores and general charges	5.50
Total cost per ton	69.30

MACARTHUR-FORREST PROCESS WITH ELECTRICAL PRECIPITATION:

500 tons per day plant, costs per ton—	Cents.
Filling and discharging	20.00
Cyanide ¼-lb. at 27c. per lb	6.75
Lime and caustic soda	3.40
White labor	10.00
Native wages and food	3.80
Fuel and power (including haulage and dynamos, and fractional maintenance and superintendence of generator and engine)	8.00
Lead	2.20
Iron	4.40
Stores and general charges (including cupellation)	6.40
Charge due to extra cost of plant	2.60
Loss of interest due to long retention of gold in extractor boxes	0.76
Total cost per ton	68.31

"The comparison," concludes Mr. Yates, "shows how little there is to choose between the two methods of precipitation in their financial aspect, and taken in conjunction with the examples of working costs previously given, dispels the idea of superiority of electricity in this respect, which has obtained acceptance with some of our general managers."

Von Gernet, in replying to Yates' paper, stoutly refutes the assertion that the strength of cyanide is determined by its extracting power, and not by its availability for precipitation; contending that 0.05% solution is quite as efficient as 0.3% solution, and that it would be used in zinc plants if it were not too weak to precipitate the

* "Transactions Chemical and Metallurgical Society of South Africa," vol. i., p. 266.

PLATE IX.

Cyanide Plant, Meyer & Charlton Mine, Transvaal.

gold on shavings. This contention is, of course, open to question. It has been a common experience with operators to find a solution of 0.2 or 0.3% strength much more efficient as a working solution than one of 0.05% strength. At one of the large works on the Rand "the dilute solution would not sufficiently dissolve the gold from the tailings . . . and the gold in the solution was not well deposited, as up to 3 dwt. per ton of solution was still left in the solution after exit from the electrical boxes." One authority* sums up this question of the comparative solvent power of strong and weak solutions, as follows: "Attempts have been made to prove that weak solutions dissolve gold (as it exists in tailings) as quickly or more quickly than the stronger solutions used by the rival process, but I am confident that I echo the opinion of the great majority of the cyanide men when I say that this contention, as applied to the treatment of ores or tailings on a large scale, has not been proved, and is in reality not an accurate statement of facts. . . . In my experience strong solutions (0.15 to 0.35% KCy) dissolve the gold from tailings in much less time than solutions of 0.02 to 0.10%. If the Primrose Gold Mining Co. had to use the Siemens-Halske process, the plant would have to be almost doubled to obtain the present extraction."

As the chief claim of the Siemens-Halske advocates hinges upon this important question, whether dilute solutions are as efficient as stronger solutions in dissolving gold, and as the balance of opinion seems now in favor of stronger solutions, it is not wholly improbable that zinc precipitation will outlive the newer method, as applied to ores and tailings, unless the use of electricity develops very distinct advantages in point of working cost and efficiency.

Recovery of Gold from Solutions.—In the best conducted plants using zinc precipitation the value of sump solutions ranges (as nearly as can be computed from varying data) between ½ dwt., or a little less, to 1½ dwt. per ton; 1 dwt. per ton, or about $1.03 per ton, is considered a fair average for sump solutions.

With Siemens-Halske precipitation the average of sump solutions is between 1 and 1½ dwt. per ton.

Extraction.—The actual gold extraction varies through a wide range, but in general averages between 70 and 75%. At the Meyer and Charlton works it reaches 72%; at the Geldenhuis Deep, 72.5%; at the Crown Reef (maximum), 80 to 85%; at the Treasury (maxi-

* G. T. M. McBride, in "Transactions Chemical and Metallurgical Society of South Africa," vol. i., p. 341.

mum), slightly over 80%; at the Robinson, 70%. All these plants use zinc precipitation.

Of the works using electrolytic precipitation, the May Consolidated (1895) reports 79%; the Worcester, 73.7%; the Metropolitan,* 55%.*

Cost of Treatment.—There is a noticeable decrease in the cost of treatment since cyanide operations commenced on the Rand, owing to various mechanical improvements, the reduction in price of cyanide, and the great increase in capacity of the more recently constructed works. The cost per ton at the Worcester Works (Siemens-Halske) is 3s., or 72c.; at the Crown Reef (zinc), ditto; at the Geldenhuis Deep (zinc), 2s. 7d., or 62c.; at the Geldenhuis Estate (zinc), 2s. 6.24d. or 60½c.; at the Robinson (zinc), 2s. 7.26d. or 62½c.; at the City and Suburban (zinc), 2s. 3.6d. or 55c.; at the New Primrose (zinc), 4s. 1d. or 98c.; at the New Heriot (zinc), 4s. 2.25d. or $1.00½.

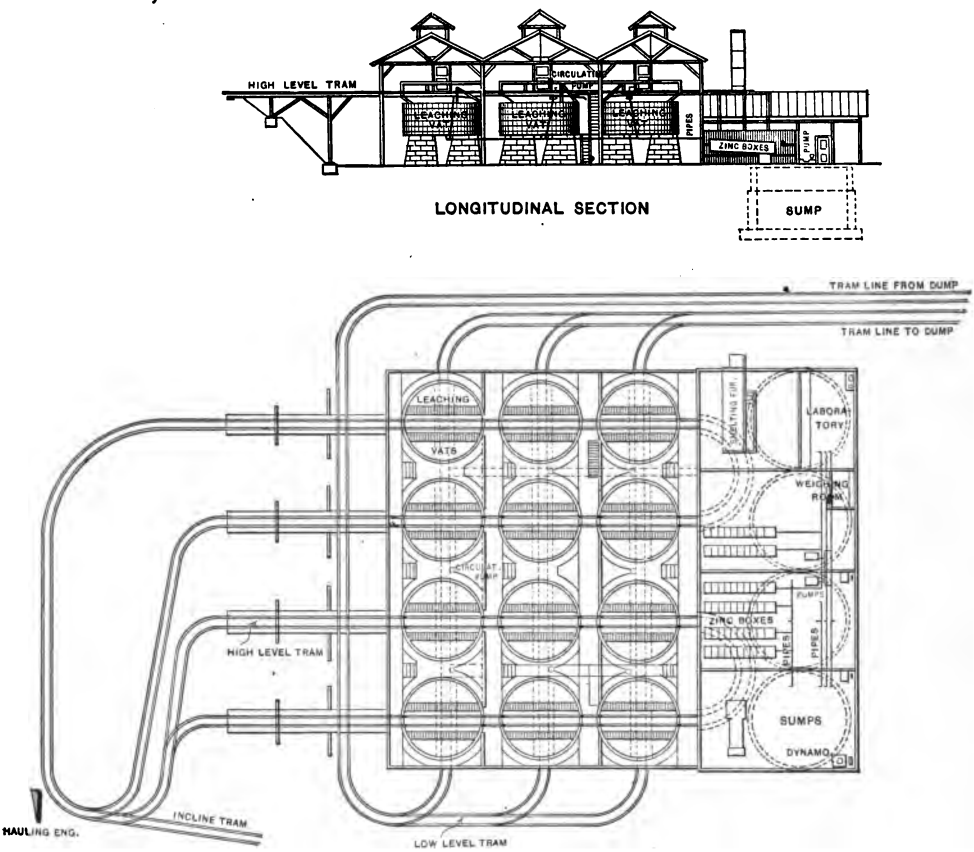
Discrepancies in Bullion Yield.—A difference between actual and theoretical extraction is observed in all the plants, a difference which is popularly attributed to errors in estimation of tonnage, errors in sampling and assaying, losses in slags, losses through leakage, theft, etc. Very little stress is laid upon errors in estimating moisture, which is really one of the most difficult things to correct in cyanide works. It will be readily seen that an error of 1 or 2% each day will make a considerable difference in the tonnage at the end of a month's run. Estimating moisture from a general sample taken during the charging of a vat is a poor practice, as no allowance is made for evaporation from this sample during the time the vat is being charged. The necessity becomes apparent of making a moisture test from every few carloads, *i.e.*, setting aside, say, 500 gm. from a single large sample, and at the end of the day combining these various lots, drying the whole, and estimating the amount of water present.

Practice at Typical Works.†—Cyanide works of Robinson Gold Mining Co. Capacity, 6,500 tons per month; tailings elevated by means of bucket tailings-wheel to intermediate vats, and sands collected by Butters and Mein distributors, then transferred to treatment vats; zinc precipitation used; value of residues, 2 dwt.; cya-

*At these works electrolytic precipitation was recently abandoned in favor of zinc.

† For much of the following account of typical works I am indebted to the "Transactions of the Chemical and Metallurgical Society of South Africa," vol. i., and to Hatch and Chalmers' "The Gold Mines of the Rand."

nide consumption, 0.5 lb. per ton; fineness of bullion, 778. (The cut represents the original Robinson Works before the changes were made for intermediate treatment. These changes consisted principally in the addition of a series of settling or intermediate vats.)



PLAN

Fig. 16.—The Robinson Company's Works, Johannesburg, S. A. R.

At the works of the Jumpers Gold Mining Co., and of the New Kleinfontein Co., Ltd., the system of direct filling is used, that is, without "intermediate" or "settling" vats. The success obtained by this system at these works is the more interesting inasmuch as "intermediate" filling has obtained the preference with Rand operators, and is used in the largest works.

At the Kleinfontein Works two pointed boxes are used, 6 ft. square and 6 ft. deep. The tailings from the battery flow into the first separator. The slimes overflow from this separator passes into the second, thereby reducing the quantity of fine material in the final overflow. The treatment tanks, of which there are 19, the 13 largest being of 200 tons capacity each, receive the pulp, of which 15% of the slimes has been eliminated in the separators. The vats, which have been previously filled with water, are allowed to overflow into a slimes-race, and by this means 10% more of the slimes is eliminated, leaving the pulp in the vats almost free from slimes. When the vat is filled an alkaline solution is run on, and this is followed by cyanide solution. The gold is precipitated on zinc. To counteract the effects of refractory foreign metals present in the ore which retard precipitation, a zinc-lead couple is set up in the boxes by dipping the shavings into a weak solution of acetate of lead before placing them in the boxes. This is said to bring about an almost perfect precipitation. The zinc-slimes are treated with acid. The average fineness of the bullion is 847; 60 silver. The extraction is 80%.

The operator who describes Kleinfontein practice (F. Cardell Pengilly) mentions as the chief advantage of "direct filling" the small plant required as compared with a plant designed for "intermediate filling." He makes no mention of the destination of the slimes, which are almost completely eliminated from the pulp before cyaniding. They are probably reserved for separate treatment; but in the meantime they are bringing in no returns and even when a system of slimes treatment is perfected, can only be worked at a considerable cost.

It is one of the weaknesses of the "direct filling" system that virtually all the slimes must be eliminated; otherwise they form layers throughout the charge, which interfere with percolation and extraction. In "intermediate filling," however, a large proportion of the slimes may be retained and at once made available—a very great advantage in view of the cost of re-handling and separately treating this material.

The Bonanza Cyanide Works.—This plant treats 3,000 tons of high-grade tailings and concentrates per month. The plant is of recent design, treating the pulp from a 30-stamp mill. A tailings plunger pump raises the tailings from the plates to the spitzluten placed over the collecting vats. These boxes separate out the concentrates and heavier sands. The collecting vats are fitted with

Butters and Mein distributors, and with slimes-discharge gates. The overflow from the latter is received by spitzkasten, which separate the very fine leachable sand and return it to the plunger pump. There are two tiers of vats, six in each tier, placed one

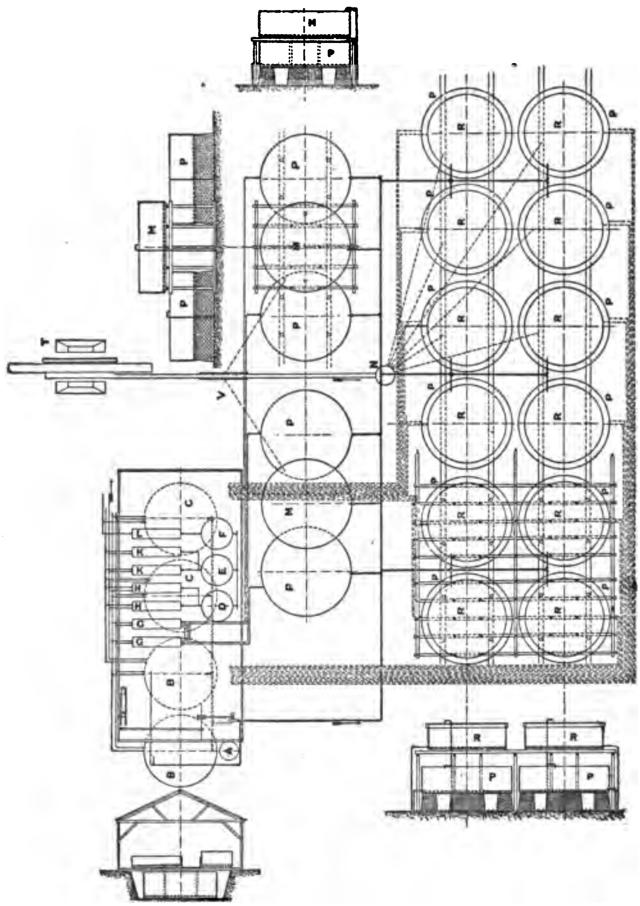


FIG. 17.—DESIGN OF A TYPICAL SOUTH AFRICAN CYANIDE WORKS.

A mixing tank, *B* strong solution tank, *C* weak solution tank, *D* strong receiver, *E* weak receiver, *F* waste receiver, *G* concentrates, *H* strong solution, *K* weak solution, *L* waste, *M* concentrate tank, *N* distributing tank, *P* treatment tanks, *R* intermediate tanks, *T* tailings whe ls, *V* spitzluten.

over the other. Each vat has a capacity of 100 tons. Two of the upper vats receive the concentrates; the other four receive the main pulp. Only 20% of the coarse material is separated

by the pointed boxes, it having been found that when more of the coarse was separated, the extraction and percolation were too slow for the best results. The standard strong solution used is 0.1%, the strength varying from 0.1 to 0.04%. The weak solution ranges in strength from 0.025 to 0.03%. The average extraction from the sands is 80.9%.

The gold is precipitated by the Siemens-Halske process. There are two strong and two weak boxes, 30 ft. long by 4½ ft. wide by 3 ft. deep, connected with a dynamo, giving 200 amperes at 8 volts pressure. Each box contains 156 iron anodes (covered with fine sacking), the total weight of iron in each box being 1,350 lb. There are 156 cathode-frames carrying 468 lb. of lead foil cut into strips 1 in. wide, thus giving a total area of cathode surface of 5,990 sq. ft. A boy is occupied cutting the lead foil into strips, and hanging these on wire frames.

The strong solution flows through the boxes at the rate of 3 tons per hour; the weak at the rate of 1 ton per hour. The precipitation is not so complete as in zinc-boxes working under the best conditions. The plant is being enlarged to 40 stamps, and will in future treat both sand and slimes. The cost of treatment per ton is at present 6s. 8.7d., or $1.61, but this will probably be much improved when the plant is enlarged.

Cyanide Works of the May Consolidated Gold Mining Co., Ltd.—This plant has undergone extensive alterations since work commenced in March, 1895. It now consists of 17 tanks arranged in double tiers, of an average capacity of 200 tons each. The tailings are delivered to the settling tanks by a 15-in. plunger pump. Siemens-Halske precipitation is used, and has given satisfaction. The solutions range in strength from 0.1% KCy for strong solutions, 0.025% KCy for weak, to 0.01–0.02% KCy for preliminary solutions. Discharging is now done with a system of mechanical haulage. The time required to clean up six precipitation boxes containing 468 lead frames is barely 12 hours, including the time for putting in fresh lead strips. The bullion obtained is about 890 fine. The full capacity of this plant is 12,000 tons per month; its cost was £27,500. The working cost during August, 1896, was 2s. 8.74d.; September, 2s. 8.8d.; October, 2s. 5.5d.

New Primrose Works.—This is one of the largest cyanide plants on the Rand, the capacity per month being 17,000 tons. The practice is typical of the best. The MacArthur-Forrest zinc process is used. The consumption of cyanide averages 0.98 lb. per

ton; zinc, 0.22 lb.; caustic soda, 0.22 lb. The working expenses average 4s. 1d., or 98c. per ton. The bullion recovered is worth 65s. per oz., or $15.60.

The old system pursued in these works was to treat the battery product without classification; the result was a saving of only 55% of the assay value. W. Bettel, the consulting chemist, recently introduced spitzluten for separating the coarse sand and pyrites. This product is now treated separately with a 0.3% cyanide solution, then discharged into vats for final treatment, where it is subjected to lixiviation for 2½ and 3 days. The extraction from these concentrates is 85 to 93%. The consumption of cyanide is 1½ lb. per ton.

The pulp from the classifiers is conveyed to intermediate tanks fitted with a baffle plate near the discharge rim for preventing a loss of coarse and medium sands with the slimes. The sand is then cyanided with solutions of various strength, 0.2% being the strongest. Vacuum pumps are used for hastening percolation and for draining the charge; and the solutions, which contain a certain amount of slimes, are passed through four filters filled with coir fiber before passing to the zinc-boxes, the slimes being collected by a filter press.

The results yielded by double treatment at the Primrose are very satisfactory, from 75 to 80% of the gold being recovered.

The New Heriot Works have a capacity of 5,000 tons per month. The tailings are divided by means of spitzkasten into three products: (1) Concentrates, averaging in value 32.93 dwt.; (2) sands, 5.03 dwt.; (3) slimes, 3.35 dwt. The concentrates and tailings are cyanided and the slimes reserved for later treatment.

The concentrates, after 13 days' treatment, yield 87.51%; the sands, after 5 days, 69.5%. The residues from the sands average 1.54 dwt. The bullion recovered averages 60s. 9d. per oz., or $14.58. The cyanide consumption is 0.84 lb.; zinc, 0.30 lb.; caustic soda, 0.20 lb.

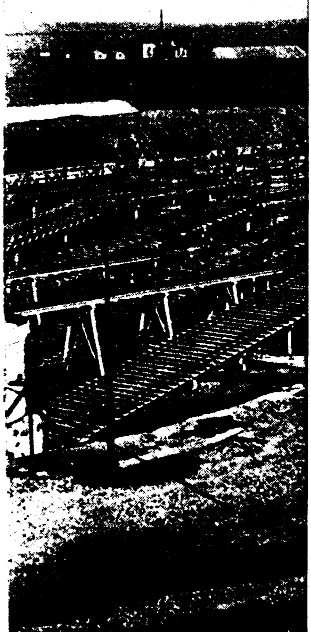
The Worcester Works.—Here the pulp passes through spitzkasten and spitzluten, separating into four products: (1) Coarse sands and pyrites, assaying 15 dwt.; (2) bulk of sands, 6 dwt.; (3) fine sand, 4½ dwt.; (4) unleachable slimes, 4½ dwt. The first product, which consists of 15% of the pulp, is treated 9 days with an 0.08% solution, leaving 1½ to 2 dwt. in the residues. The second product constitutes 50% of the pulp, is treated for 5 days, leaving 1 to 1¼ dwt. in the residues. The third product amounts

to 10% of the pulp. The residues average 1 dwt. The fourth product amounts to 25% of the pulp, and is not treated at all.

The electrolytic process is used. The consumption of cyanide is ¼ lb. per ton, and the cost of treatment 3s. or 72c. per ton. The capacity of the plant is 3,000 tons per month.

New Works of Simmer and Jack Gold Mining Co.—The company's old works have been converted into a slimes plant, and new works erected, about a quarter of a mile distant from the old, for double treatment of the sand pulp from the adjacent 280-stamp mill. The two plants now form a complete system, the slimes overflow from the separators at the new plant being conveyed to the old for treatment. The mechanical arrangements at these extensive works—probably the largest in the world—and the operations carried on, are based upon the most improved and latest methods of practice, as applied to mill tailings. The plant has a maximum capacity of about 1,400 tons per day. It consists, briefly, of 30 "intermediate" vats 12 ft. deep and 24 ft. in diameter; 30 leaching vats, 10 ft. deep and 30 ft. in diameter; 16 precipitation boxes (Siemens-Halske) 30 ft. long by 4 ft. 6 in. wide; three tailings-wheels, 42 ft. in diameter, and pointed boxes for separating the coarser material from the slimes. The tailings are conveyed from the mill about 600 ft. to the wheels which elevate them to the separators. The slimes overflow from the separators is carried to the old works in a launder; the coarse material, with a small admixture of slimes, runs by gravitation to the Butters and Mein distributors, and thence to the different intermediate vats. Here the material is subjected to a short cyanide treatment, and finally shoveled out through bottom-discharge doors (six for each vat) into trucks which run on double tracks under each line of vats. The trucks are hauled up an inclined trestle to the leaching vats, where the tailings receive a second and longer treatment. The leaching vats are likewise provided with six bottom-discharge doors each.

From the storage tanks the solution is pumped to the leaching vats by means of four centrifugal pumps with 4-in. discharge. The solutions passing from the leaching vats are conducted to intermediate tanks through 1½-in. pipes. From the intermediate tanks they are pumped up to the precipitation boxes, and thence to the storage tanks. It is estimated that when this plant is running smoothly the cost of treatment per ton will not exceed 2s. (48c.).



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CHAPTER XIV.

EXEMPLIFICATIONS OF PRACTICE: AUSTRALIA, NEW ZEALAND, AND INDIA.

AFTER South Africa the cyanide process finds its most important applications in Australia and New Zealand. According to G. T. Beilby * the total production of gold by the cyanide process in 1897 was as follows: Africa, 825,000 oz. of bullion; Australia, 308,000; New Zealand, 263,000; United States, 190,000; India, 18,800; Mexico, 10,200; other countries, 5,000—total, 1,620,000, equivalent to 1,215,000 oz. of fine gold. These figures are only approximate, but they serve to give an idea of the relative importance of the cyanide process in the various countries of the world.

In Australia the cyanide process finds its chief application in Queensland, especially in the Charters Towers district, where in 1897 there were 57 cyanide works in operation and several in course of construction. The total gold production of the district in that year was 356,658 oz., of which 107,415 oz. were obtained by the cyanide process.† The cyanide process is also used in the Gympie and Croydon districts in Queensland. There are also numerous cyanide works in New South Wales, Victoria, South Australia, and Western Australia.

Outside of Queensland the most important application of the process appears to be in Western Australia, where it is now the consensus of opinion that the telluride gold ores of Kalgoorlie and Coolgardie must be treated by cyanide lixiviation, preceded by a preliminary roasting. Several important new developments in the treatment of this class of ore have already been made at various works in this district, noteworthy among them being the treatment of slimes by the filter-press process as practiced at the Lake View Consols mine, and the metallurgists of Western Australia bid fair to take the lead in the treatment of this class of ore. The practice

* *Journal of the Society of Chemical Industry*, Feb. 28, 1898.
† "The Mineral Industry," vol. vi., p. 272.

at Kalgoorlie has not yet crystallized, however, and no technical accounts are now available. The average cost of cyanide treatment of sands at four mines is said to be 5s. 3d. per ton. It appears that the slimes treatment by the filter-press process can be done at 8s. to 10s. per ton.* The proportion of slimes varies at each mine, but by dry-crushing the minimum proportion is secured; what proportion (if any) it is necessary to separate in the treatment of the telluride ores is still entirely unknown.

The filter press treatment of slimes was developed in Western Australia largely through the scarcity of water, which excluded the adoption of the South African practice of coagulation with lime and washing by decantation, since by the latter method the discharged slimes still contain 50% of their weight of weak cyanide solution, besides which there is the increased loss of water by evaporation from the larger quantities in circulation. At Hannan's Brownhill mine at Boulder, Western Australia, the ore is crushed dry and then divided by a pneumatic separator into sands and slimes. The former are leached in vats in the ordinary way; the latter are agitated with a 0.3% cyanide solution for six hours in a vat, after which the thin mud is forced into a filter press and squeezed, whereby the gold bearing solution is separated, the cakes being washed by subsequently forcing water through the press. The whole operation of filling the press, leaching and emptying, occupies from 1½ to 2 hours. At the Lake View Consols mill the wet tailings from the battery are partially settled, and the slimy liquor is treated in a set of 10 filter presses which give an 85% extraction with $10 slimes.†

At Hannan's Brownhill mill the mud from the agitators is forced into the filters at a pressure or 60 to 80 lb. per sq. in. by means of compressed air. The filtrate flows in a constant stream through the zinc precipitation boxes to the sumps. The cakes having been washed the press is opened and the cakes are allowed to drop into a truck standing under the press. The filter presses are of the distance frame, flat plate, and four-eyed type, each press having 20 chambers, which form cakes 28 in. square by 3 in. thick. The pressed cakes carry about 20% moisture and weigh about 130

* I am indebted to Mr. R. P. Rothwell, editor of "The Mineral Industry," for much of the information concerning the cyanide process in Australia, India, etc. He has permitted me to anticipate "The Mineral Industry," vol. vii., by submitting to me several manuscripts touching upon this subject, particularly one by Mr. Walter Renton Ingalls,

† Edward S. Simpson, "Transactions of the American Institute of Mining Engin ers," February, 1898.

lb. per cu. ft., from which figures it appears that each press treats about 1.25 tons of dry slime per charge.

Experiments carried out on 3-in. cakes show that the washing is very complete and that uneven percolation does not take place. Filling the press from the receiver (in which the mud is accumulated from the agitators) occupies 15 minutes, washing the cakes in the press requires 19 minutes, and discharging and closing the press 16 minutes, a total of 50 minutes. The amount of solution required in the agitators to reduce the pulp to such consistency that it can be readily forced into the filter presses is about 1.5 times the weight of the slime. The amount of wash water required is very small, but no figures as to this are given in the paper from which this account is taken.* In the treatment of slimes by the decantation process in South Africa from 8 to 10 tons of liquid are required for the treatment of 1 ton of slimes.

Outside of Western Australia there does not appear to be much novelty in the treatment of ores by the cyanide process in Australia, and there is very little literature descriptive of the works and methods. Mention should be made, however, of the treatment of slimes by Deeble's patent agitator, which, according to W. B. Gray in "The Transactions of the Australasian Institute of Mining Engineers," vol. v., is accomplished successfully at the South German mill at Maldon, Victoria. Here the agitation vats, which are 18 ft. in diameter and 4 ft. deep, are first filled with 2 ft. of sump solution, and the agitator then set in motion, after which slimes are charged to the amount of 15 tons, which brings the level of the liquor to within 3 in. of the top of the vat. A charge of 80 lb. of lime is then added, partly as an alkalizing agent, but chiefly to help in the subsequent settlement of the slimes. A proper quantity of cyanide is then put into the vats to bring the solution up to the strength of about 0.15% and the agitation is continued for at least 30 hours, at the end of which time the agitator is stopped and raised out of the body of the slime, which is allowed to settle. This takes from 8 to 16 hours, according to fineness. As the settling proceeds the clear supernatant liquor is drawn off gradually, by depressing the outlet gate, and run into a filter vat where any fine slimes which may have escaped are arrested. The drawing off of the first liquor having been completed, the gate is

* William McNeill, "Filter Press Treatment of Gold Ore Slimes at Hannan's, West Australia," in "Transactions of the Institution of Mining and Metallurgy," vol. vi., 1897-98. p. 247, *et seq.*

raised, the agitator set in motion, and the first sump solution wash is run on. Agitation is continued 30 minutes and the first wash is then drawn off. A similar process is followed with a second sump solution and a final water wash, the quantity of the latter being gauged so as to keep the total quantity of solution in circulation constant. Finally the slimes are sluiced out. The gold is precipitated from the solution by means of charcoal; this is prepared by a Dunn's charcoal crusher and afterward cleansed by washing with water. There are 40 filters, which number is required for the treatment of about 400 tons of tailings per week. They are arranged in eight rows of five in a row, the flow of liquor being distributed equally to each row. Each filter has a capacity of about 560 lb. of charcoal. Experiments have shown that the first filter precipitates 45% of the gold, the second 25%, the third 15%, the fourth 9%, and the fifth 5%. Thus 99% of the gold is recovered, while the average assay of the outflow is only 2 grains per ton. These results are with a flow of about 200 gal. per hour. Mr. Gray considers that precipitation by charcoal is cheaper and better than by zinc. The cost of charcoal is about 2s. per filter, or 2.75d. per oz. of gold saved. Under the most favorable circumstances 1 lb. of zinc at 6d. per lb. will be consumed for each ounce of gold, while other expenses of recovery are greater than with the charcoal method. The gold from charcoal precipitation is of greater fineness than from zinc. Charcoal as a precipitant also has the advantages, it is claimed, that it will throw down the gold from the weakest solution of cyanide, and that it does not contaminate the solution. This is an application of the Johnston process.* It is analogous in all respects to the carbon precipitation of gold from solution in chlorine water, which has been employed on a large scale at several works, but has generally been abandoned in favor of precipitation with hydrogen sulphide. The carbon carrying gold constitutes a very bulky mass, troublesome to handle in recovering the gold, which is done by burning off the carbon in cast-iron pans wherein there is considerable danger of loss.† There is reason to believe that Mr. Gray underestimates the merits of zinc precipitation and overestimates the cost. However, the high position which he occupies in cyanide metallurgy in Australia, being the pioneer in the introduction of the process on a large scale and manager of

* "The Mineral Industry," vol. iv., p. 342.

† Walter Renton Ingalls, "Progress in the Metallurgy of Gold and Silver," in "The Mineral Industry, vol. vii.

the South German Works, which are in some respects among the most complete in Australia, gives weight to his opinions.

At the South German mill, where the above described process for the treatment of slimes is carried out, the ore is crushed wet by stamps and the gold amalgamated on plates. The tailings are concentrated for separation of the pyrites, which is treated by chlorination, blanket strakes being used for this purpose. The tailings from the blankets are leached with cyanide solution. There are four lixiviation vats, each capable of treating 110 tons of tailings at a charge. They are 9 ft. deep and 25 ft. in diameter and are built of wood, with discharge doors in the side, which are large enough to allow the wagons to run in on loose rails so that they can be filled inside the vat. The time of treatment per charge is 96 hours, counting from filling to filling. The strong solution contains about 0.1% of cyanogen, and the weak solution about 0.06%. The tailings are charged into the vats by Butters distributors. The slimes formerly wasted are now treated by the Deeble system above described. The gold is precipitated from the solution by means of charcoal. The total cost of treatment is about $0.63 per ton, of which $0.32 are for cyanide.*

At the Day Dawn mine in Western Australia the Sulman-Teed bromo-cyanogen process has been applied to the treatment of battery tailings. The bromo-cyanogen solution is made up in a special tank to about the strength of 7 or 8%. This is then mixed with the cyanide solution in a fixed proportion. The time required for extraction is 14 to 15 hours. The consumption of cyanide is 4 oz. per ton; of bromo-cyanogen, 1.75 oz. per ton. The percentage of extraction is said to be about 90. A very interesting feature in connection with this process is the precipitation of the gold by means of zinc-fume. "The gold is precipitated from the solution in a special apparatus known as the precipitation cone, which is made of galvanized sheet-iron, about 5 ft. in diameter at the top and of similar depth. Two cones are placed in series, one about 2 ft. above the other. Around the top of each there is a circular gutter to collect the overflowing liquor which runs from the first cone to the second and from the second to the sump for gold-free solution. The gold-bearing solution enters the first cone through the bottom and is distributed by means of a small perforated, inverted cone-nozzle, inside of which there is a valve arrangement to

* James Mactear, Notes on the South German Mine, Maldon, Victoria, "Transactions of the Institution of Mining and Metallurgy," vol. iv., 1897-98, p. 43.

prevent any backward flow of zinc emulsion when the cones are stopped. The bottom is also provided with a three-way cock, which either permits the inflow of liquor for precipitation or shuts this off to permit the discharge of the gold-impregnated zinc fume, as desired. The zinc fume is added in the form of an emulsion through a small funnel in the center of the cone, which is enlarged at the lower end for the purpose of producing a sort of vortex chamber at the bottom of the zinc cone, which insures a more thorough mixture of the zinc-fume emulsion with the inlet liquors.

"Assuming the cones to be full of gold-free liquor and the gold-bearing liquor from 100 tons of tailings to require precipitation, the routine is as follows: The gold-zinc fume from cone No. 1 is removed for clean-up; the partly used gold-bearing fume from cone No. 2 is transferred to No. 1 and a charge of about 12 lb. of fresh zinc fume in an emulsion with water is run into cone No. 2. The gold-bearing liquor is then run through the series at the rate of 400 to 600 gal. per hour. The zinc cone automatically grades the zinc particles so that the heavier are in the richest gold-bearing stratum, and the liquor in rising through the cone, as it becomes poorer in gold contents, meets finer and finer suspended particles of zinc; but inasmuch as these become weighted with gold after a time there is a tendency for them to be withdrawn from the top zone of slow upward travel, and it is consequently advisable to add every hour or two about 1 lb. of dry fume in emulsion through the central funnel" *

It is claimed that the overflow solution from the cones at the Day Dawn works does not carry more than 10 grains of gold per ton. In general, however, the peculiar advantages claimed for the bromo-cyanogen process have not been satisfactorily demonstrated in practice.

The Cyanide Process in New Zealand.—It is claimed that Karangahake was the first place in the world, outside of Glasgow, where the cyanide process was applied experimentally to ores. The trial was made on Woodstock ore in July, 1889, under the supervision of John McConnell, who came from Glasgow to introduce the process. The results were satisfactory. The Woodstock company at once erected a small cyanide plant which was run in connection with a 10-stamp battery. Later on a 40-stamp battery was put up, and large cyanide works fitted with the latest improved

* "The Mineral Industry," vol. vi., p. 346.

appliances. Since then the difficulties experienced in treating New Zealand ores by the more common metallurgical processes have been in great measure overcome by the cyanide process.

The ore typical of that found in the Hauraki fields, where cyanide is most successfully applied, ranges in value between £4 and £5 per ton, carrying about 3 oz. silver to 1 oz. gold. In the deeper levels the gold, as a rule, is finely divided; in the upper levels it is often coarser. The Hauraki ore is described as a hard, whitish-gray quartz, singularly free from refractory bases, and containing only a small quantity of iron oxides. The New Zealand ores, however, are not of so uniform a character as they are popularly thought to be. In a recent paper * read before the New Zealand Institute of Mining Engineers, the writer says: "There are few, if any, mines on the Hauraki gold fields where the ore does not contain a fair percentage of the gold described [coarse gold, which is practically unaffected by cyanide], and in a great many cases part of the loss in treatment of ore by the cyanide process may be attributed to this cause. Then again, the cyanide process is undoubtedly defective, under ordinary circumstances, for the treatment of ore containing pyrites, especially where the pyrites are rich in precious metals. Although such ore may be all stamped through a 40-mesh screen, and a large proportion of it will pass through a 90-mesh screen, still each particle of pyrites contains a smaller particle of gold, which is so completely enveloped as to be proof against the attack of cyanide, regardless of either strength of solution or time that may be brought to bear on its treatment. Wherever it is possible to do so, concentration should come in before cyanidation, because it relieves the ore of the base minerals that are leading factors in giving trouble and causing loss in the working of that process. There is not a single plant on the Hauraki gold fields capable of dealing successfully with refractory ore, a want which will become more and more apparent as depth is attained in the workings of the principal mines."

At Waihi, Kuaotunu, and the Whangamata Proprietary mines, the gold exists in a free and exceedingly fine state of division; at the mines of Waitekauri and Karangahake it is somewhat coarser; it is coarsest in the Thames and Coromandel districts, and accompanied by pyrites and other more deleterious sulphides.

In view of these facts, the present status of the cyanide process in New Zealand may be summed up as follows: It has proven con-

* Extract in "New Zealand Mines Record," vol. i., p. 399.

spicuously successful in the direct treatment of certain ores containing finely divided gold, as at Waihi and the Crown mines; but it has given trouble in the treatment of more complex material, where the pyrites are not nearly so amenable to cyanidation as in South Africa and elsewhere. Metallurgists are aware of the fact that the ore, even in the same mine, cannot be relied upon as being of uniform character; rock from different levels requires different treatment. The necessity is apparent of erecting plants capable of dealing with the various classes of rock as they appear. The more recent tendency has been to remodel the older crushing and cyaniding works to meet these fast-developing conditions. So far as the cyanide process is concerned, mining matters may be said to be in a state of transition, though the beginning of a reaction in favor of wet-crushing is noticeable. Dry-crushing has been abandoned in a few of the representative works, and this change seems likely to be adopted by other companies.

In the earlier days of New Zealand mining the ores were treated by copper-plate amalgamation, which, however, owing to the fine state of division of the gold in the then exploited ledges, saved only a small percentage. This was followed by dry-crushing and pan-amalgamation, which saved about 60%. The cyanide process is said to have raised the extraction to between 80 and 90%, and in some instances to over 90%.

Three typical New Zealand works are those of the Crown Mines Co. and the Woodstock Gold Mining Co., at Karangahake, and the Waihi Gold Mining Co.'s works at Waihi and Waikino.

The Crown Mines.—The former practice at the Crown Mines was to dry the ore in brick kilns, from which it was passed into a hopper, whence it was delivered to Challenge ore-feeders. The ore was dry-crushed by stamps, from which it fell into a chute, whence it was conveyed directly to cyanide vats. The cyanide plant (recently increased by the addition of 13 steel vats) consisted of 24 wooden vats, 11 ft. long and 9 ft. wide, and 3 ft. 9 in. deep; also 14 agitators, 8 of which are 5 ft. deep by 4 ft. 9 in. in diameter, and 6 are 6 ft. deep, and 5 ft. 6 in. in diameter. The agitators, however, are seldom used, the preference at present being given to cyaniding by percolation. There are, besides, zinc precipitation boxes and three concrete sumps, 15 ft. by 12 ft. and 6 ft. deep. The cut represents the general features of the old plant, where the practice has recently undergone interesting changes.

Wet-crushing has been adopted, it having been found after care-

ful experimentation with both the wet and dry methods that wet-crushing yields a higher percentage of bullion, and that the treatment is more economical and pleasanter for the workmen. The mill has been enlarged to 80 stamps.

With the present ingenious system the ore is crushed in the presence of a dilute solution of cyanide (about 1 lb. to the ton of water) instead of water. The pulp is run into cyanide-treatment vats, where it is subjected to stronger cyanide solutions; the overflow of slimes is conveyed to separate tanks, where the solution is decanted off into zinc-boxes and thence conveyed back to the battery. The slimes are partially dried by means of vacuum pumps, and then sluiced out. The sand charges are sluiced out over amalgamated copper plates. This system is said to entail a greater consumption of cyanide than in the dry-crushing process; but this is counterbalanced by distinct advantages, such as doubling the capacity of the mill, and saving the great expense previously incurred in drying the ore.

The Woodstock Works are about to be converted into a wet-crushing plant, the proposed method of working being similar to that adopted at the Crown works. The following is an interesting account of the new system of treatment:*

"The only difference between the use of a dilute solution of cyanide in the mortar-boxes and ordinary wet-crushing with water is that the solution is made to describe a circle, while the water, after doing its work, is usually allowed to run to waste. To begin with, the solution is made up generally in the sumps, from whence it is pumped to the stock solution-vats that are located on a high elevation so as to admit of the solution flowing automatically through a pipe into the mortar-boxes, which, of course, are charged simultaneously by the self-feeders with raw ore. After stamping the pulp, solution and finely reduced ore pass through the 40-mesh screens and drop on an apron, which is fixed in front of the mortar-boxes. Going away from the boxes this apron contracts and terminates in a launder, which conveys the pulp to the vats, over which it is distributed by means of cross launders. This system of filling vats is somewhat primitive, and later on it will be improved upon by the introduction of automatic distributors. When No. 1 vat is full, the pulp is directed into No. 2 vat, and thence into No. 3, or No. 4 vats, as the case may be. Within two hours the slimes are settled sufficiently to permit

* "New Zealand Mines Record," vol. i., p. 294.

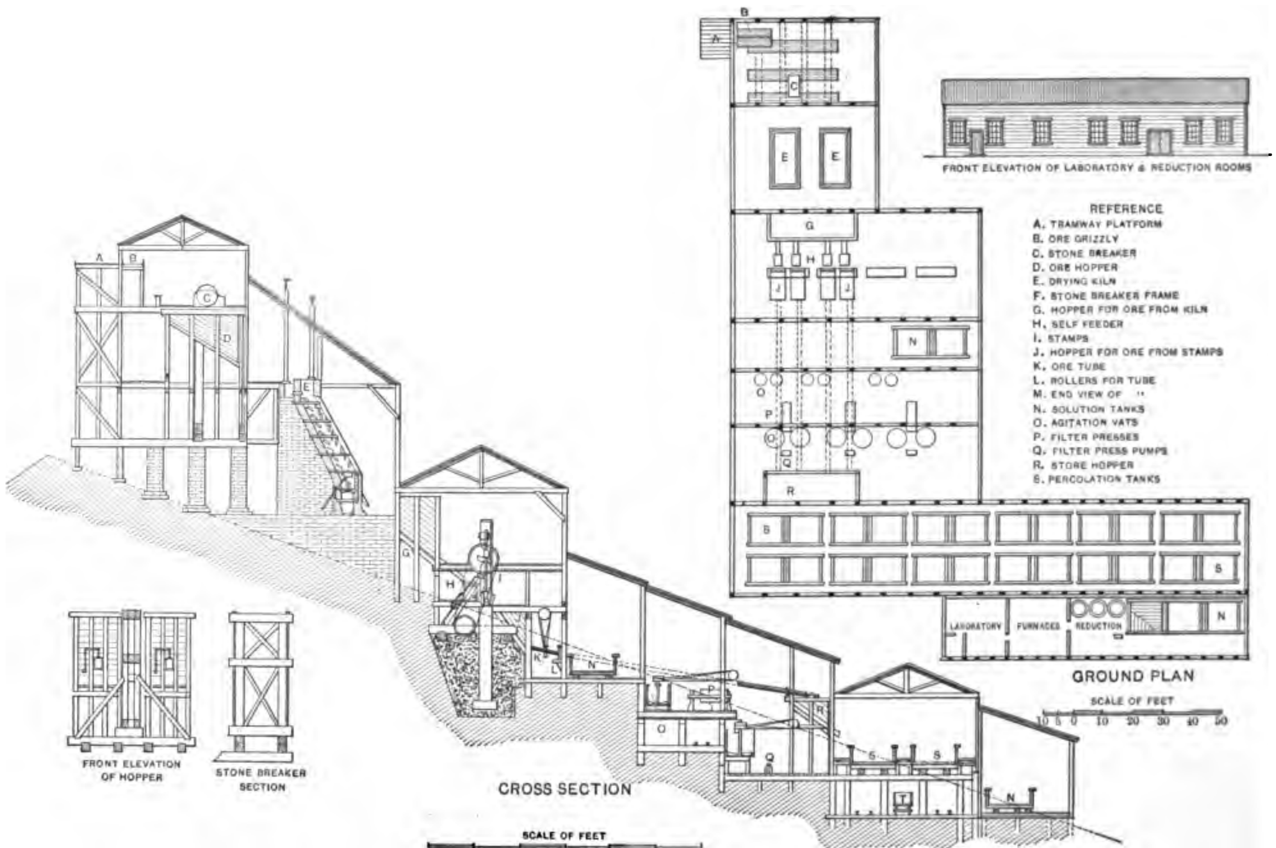


FIG. 18.—CYANIDE WORKS OF THE NEW ZEALAND CROWN MINES CO.

of the solutions being siphoned off No. 1 vat, from whence they are conveyed by a launder to the sumps, and from there they are pumped up to vats fitted with filters for eliminating any slimes that may be suspended therein. Once through the slime vats the solutions find their way down pipes, arranged for the purpose, to the extractor boxes, where the gold and silver are deposited on the zinc shavings, and then they pass into the storage sumps, where they are rehabilitated and subsequently pumped up to the stock solution vats to pass through the same routine again and again. The time taken to fill a vat with pulp varies considerably, according to the character of the ore, and the volume of solution used in the mortar-boxes. The quantity of solution brought to bear on wet-crushing varies from 2½ to 3½ tons per ton of ore, and it is necessary to turn the pulp into the vats three or four successive times, and settle and siphon off the solutions during the intervals before the requisite quantity of ore, say 30 tons, is deposited in each vat. With this system of treatment two important factors in the successful working of the cyanide process are brought to bear on the ores—namely, agitation and oxygen. In fact, fully 40% of the bullion value is extracted before the ore leaves the mortar-boxes, and by the time the pulp reaches the vats the solutions have absorbed 50% of the assay value, on the average. There is but little difference in the time required for treating ore, either wet or dry, and the waste solutions or water-washes are put on and withdrawn by the vacuum pumps, the same way in both instances. There are no scientific or chemical problems to be solved in connection with the working of this process. It is a purely mechanical business from start to finish, and the system under which it is worked, by siphoning the solutions off the top of the charge, after allowing the slimes to settle, has been used in pan-amalgamating plants for many years. The success of the wet process, when applied to the treatment of Woodstock ore, may be attributed solely to the fact that the ore is crystalline, free from clay and earthy matter of every description; consequently, the percentage of slimes formed in wet-crushing is much less than in the case of dry-crushing, because the ore is removed from the mortar-boxes by the dilute solution almost as soon as it reaches the required degree of fineness to pass through the screens. This accounts for the rapidity with which the water-washes go through the vats containing wet-crushed ore as compared with the vats that have been filled with dry-crushed ore, because the latter is stamped so fine

before escaping from the mortar-boxes as to form a high percentage of slimes, and these invariably retard the work of water-washing the ore charges. Briefly stated, all other things being equal, the success of the wet system depends entirely on the question of slimes. and these are produced in wet-crushing mainly through the presence of clay or other earthy matter that may be inseparable from the ore before treatment."

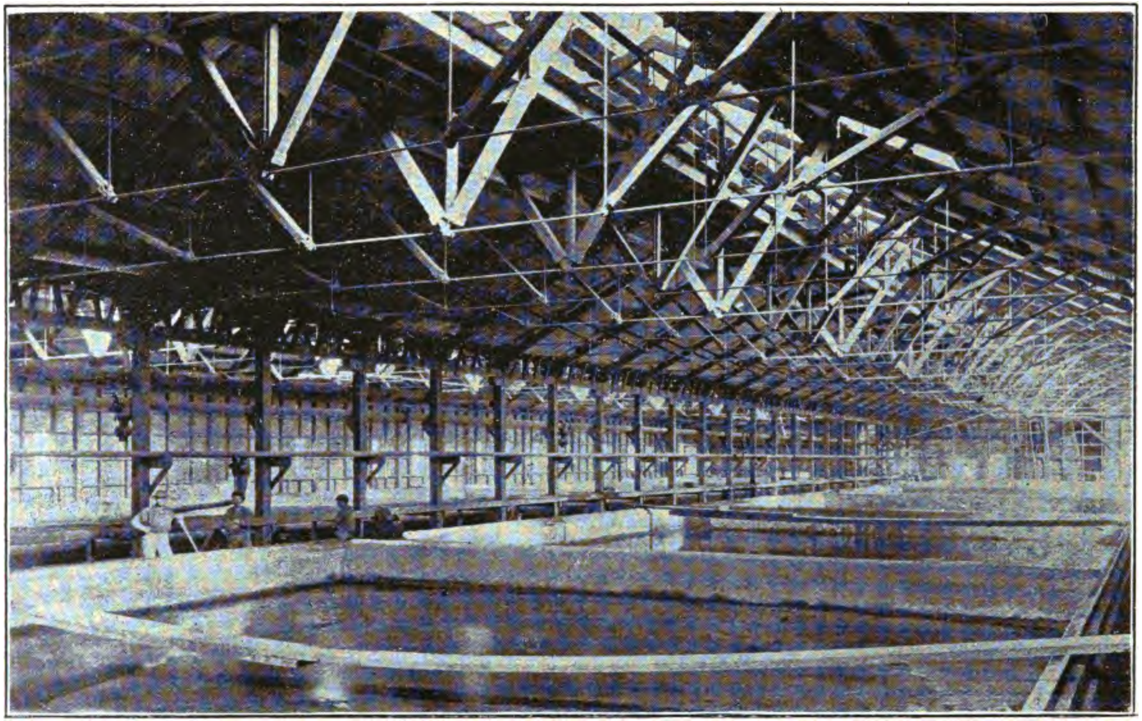
The Waihi Cyanide Works.—At the Waihi Co.'s Works dry-crushing and direct cyaniding are still practiced, this method having superseded unsatisfactory pan-amalgamation. The ore is very similar to that of the Crown mines, being a clean quartz, carrying free and finely-divided gold. The ore is dry-crushed by stamps, and passed through a 40-mesh screen. The percolation and washings are done in four days, the crushed ore being found perfectly permeable to cyanide solutions.

The ore is first dried in large open kilns excavated in sandstone. These kilns are charged with ore and wood in alternate layers, the cost of firewood being 37½c. per ton of ore. From the kilns the rock is put through a rock-breaker, from which it falls into a hopper, which automatically feeds it to the mortars. From the stamp mortars the ore is conveyed by an Archimedean screw to the dust bin, from whence it is lifted with a bucket-elevator, and discharged into a belt-conveyor, which in turn discharges into a hopper 110 ft. long, provided with 20 doors for discharging into trucks. The trucks run on rails over the vats, and are provided with hand-traversing gearing, which enables them to be dumped at any part of the vat. The vats are 22 ft. 6 in. in diameter, and 4 ft. deep. The precipitation boxes are 16 ft. long, 2 ft. deep, and 17 in. wide, and are divided into 12 compartments.

New works are now in operation at Waikino, a short distance from the old, consisting of 90 stamps and an extensive cyanide plant, with 10 concrete rectangular percolating vats of a capacity of 250 tons each. This increases the Waihi Co.'s batteries to 190 stamps, making it the largest milling and cyaniding plant in New Zealand.

The Moanataiari Co.'s Works, situated at Thames, are of recent construction, and in point of completeness and modernness of equipment, among the finest in New Zealand. The plant consists of a 60-head battery, two rock breakers, grizzlies, automatic feeders, 24 vanners, 21 Berdan pans, a complete cyanide plant for treating vanner concentrates, 9 Cornish buddles for concentrating the tailings from the vanners, and all up-to-date appliances for assay-

PLATE XI.



Cyanide Vats of Waihi Gold Mining Company, Waikino, New Zealand.

ing and retorting. The cyanide works consist of three steel vats, 20 ft. in diameter, and 7 ft. deep, each provided with two Butters bottom-discharge doors, with a capacity each of 200 tons of concentrates; two zinc extractors, each 15 ft. 6 in. in length; and three concrete sumps, 50 by 11 by 6 ft. over all. Wet-crushing, concentration, and the cyanidation of the concentrates are the interesting features of the practice at these works. The cyanidation of concentrates is comparatively new in New Zealand.

An agitation plant for the treatment of concentrates was erected by Dr. Scheidel at the works of the Sylvia Co., Tararu, Thames, and operated with success, the extraction of gold averaging about 90%.

Beside the plants mentioned, there are those operated by the following companies: The Waitekauri, Union Waihi, New Zealand Talisman, Tararu Creek, Mariposa, Try Fluke, and others.

The Cyanide Process in India.—Laurence Pitblado states* that in the Kolar field there are at present six cyanide works in operation. The ores of the field are very simple, consisting mainly of pure quartz, with only a small percentage of pyrites. The material treated is tailings from the stamp mills. Those first worked in the Mysore plant averaged 4.5 dwt. gold, and yielded 65%, with an average consumption of 1 lb. cyanide per ton. In 1897 a month's test with 40-mesh screens in the batteries gave the following result: 90.65% amalgamated in batteries and on plates; 74% of the value in the tailings recovered by cyanide lixiviation, making a total extraction of 97% of the ore as delivered to the mill. The cost of cyaniding at the present small plant of the Mysore company is 2s. 10.5d. per ton, exclusive of royalty and depreciation, but in the new 4,000-ton plant which is being erected alongside the heap of accumulated tailings that is to be worked, it is believed that the cost will not greatly exceed 2s. At the Champion Reefs mill, where 20-mesh screens are used in the batteries, the extraction from the tailings is about 56%, with a consumption of 1 lb. cyanide per ton.

In refining the precipitate in this district it is first passed through a 30-mesh screen, drained, dried, roasted with or without a small percentage of nitre, and fused directly in plumbago crucibles. At the Mysore works the precipitate is retorted before roasting, yielding about 100 lb. of mercury per month. The presence of mercury in the zinc-boxes generally leads to the production of much floured

**Journal of the Society of Chemical Industry*, Feb. 28, 1898.

and brittle zinc. In fluxing and smelting directly the retorted slimes the bullion assayed 56.4% gold, 3% silver, 2.4% lead, 19.6% copper, 18% zinc, and 0.1% nickel. The slag contained a good many shots of metal. In roasting with nitre a slag freer from prillions was produced, and a bullion assaying 49.5% gold, 4.6% silver, 4.29% lead, 38.21% copper, 2.10% zinc, and 0.22% nickel. These results being unsatisfactory the following practice was adopted at the Mysore works: The retorted and dried slimes are mixed with 10% nitre and roasted at a bright red heat. When cold they are boiled with dilute sulphuric acid (1 : 2) which dissolves the copper. The dried and washed product is fluxed with about 35% borax, 15% soda, and 10% sand, giving a slag free from shots of metal and a bullion assaying 81.3% gold, 6.9% silver, 2.71% lead, 6.78% copper, 0.4% zinc, 0.12% nickel.

CHAPTER XV.

EXEMPLIFICATIONS OF PRACTICE: UNITED STATES.

It is said that in the United States there are at present about 40 cyanide plants in operation. The practice in different localities varies greatly, in accordance with the various character of the ores dealt with. The most complex problems in cyaniding have probably arisen in connection with the ores of Cripple Creek, Colo. At Mercur, Utah, the ores are treated direct; but fortunately they are, with few exceptions, of a character which has not called for the solution of any complex chemical or mechanical problems. At Bodie, Cal., Harquahala, Ariz., and at the Eureka Works near Carson, Nev., the simple problem of dealing with mill tailings has been successfully met. At Marysville, Mont., the low gold value of the tailings and certain difficulties with precipitation seemed at first to put the matter of their economic treatment out of the question; but a series of practical tests finally led to the erection of large works, which have been profitably operated.

Utah.—The Camp Floyd district in Utah enjoys the distinction of being the first place in America where the cyanide process was successfully applied on a large scale. The history of the Mercur mine (located in Tooele County, Utah, about 50 miles southwest of Salt Lake City) is extremely interesting, in view of the subsequent development of the property by means of the cyanide process. In 1890 the mine was a mere prospect hole, with great bodies of low-grade ore in sight. The property was owned by a company from Nebraska, and incorporated under the laws of Utah as the Mercur Gold Mining and Milling Co. In 1891 a pan-amalgamation plant was built, at a cost of $30,000, but only about 20% of the gold could be saved, at a cost of something over $4 per ton for milling. After the failure of this project, the company, says Mr. Trimmer (the foreman of the cyanide works, who has kindly furnished me with information about Mercur), was facetiously dubbed by local mining men, the "Nebraska Farmers' Sinking

Fund Association." "These men had spent nearly all their little fortunes on the mine without realizing one dollar profit. Indeed, it looked as if they would have to give up their mine, and return to farming." In the meantime, however, they heard of the success of the cyanide process in South Africa, and accordingly commenced a series of experiments at their small assay office in Salt Lake City. The results were successful, and it was decided to mortgage the mine to raise money to build a cyanide mill. In this way a plant was erected with a capacity of 10 tons per day. This small mill consisted of one Dodge crusher, one set of Wall rolls, and five 7-ton tanks. Eight men were required to run the mill, and six the mine. After the company found a suitable method of working the ore, capital soon came to their assistance, and the mill was enlarged the same year to 50 tons per day capacity. The mine was further developed, and larger ore bodies were discovered. In 1893 the mill was enlarged to 100 tons daily capacity; on September 23d of the same year a dividend was declared of $25,000. In 1895 a local railroad was completed (the Salt Lake and Mercur), with the owners of which the Mercur company made a contract for hauling ore from mine to mill. This distance of several miles had previously been covered by teams. The teams were taken off, and the mill enlarged. An ore-bin was built of large capacity; a larger Dodge crusher was added to meet the demand called for by the ore-hauling contract. In July, 1896, the whole plant was remodeled; a 500-ton Gates crusher was set up, the daily tank capacity increased to 325 tons, and the number of ore tanks was increased from 31 to 52.

The Mercur ore has been described as follows:* "Silica, in a form similar to silicious sinter, or geyserite, characterizes the ore. Cinnabar is most abundant in this rock, and forms beautiful incrustations in the cellular varieties. Wherever found in the district, cinnabar is considered a sure sign of gold. Orpiment and realgar occur in large quantities in some of the ore. There is about as much iron as is usually found in impure limestone and clay. Barite and gypsum occur more or less crystallized; also masses of limestone are found mineralized in rings, the outside assaying from $6 to $8, and the center a trace in gold. No free gold is visible in the ore even with a microscope. One remarkable feature is the absence of silver. The average of the ore milled is kept close to $12 per ton."

**Mining and Scientific Press*, May 23, 1896.

The ore is delivered by the railroad to an ore-bin 40 ft. long, 20 ft. wide, and 20 ft. deep. It is crushed in a Dodge crusher, from which it passes to a set of Wall's corrugated rolls, and is finally trammed to the cyanide vats, after being crushed to 1 in. mesh or less. The vats are 12 ft. 8 in. in diameter, and hold 15 tons. They are made of tank iron, with redwood bottoms. The filter cloth on the false bottom is burlap, and lasts from four to six weeks. From the tanks the solution is conveyed to a collecting tank, from which it is pumped by Blake single-acting pumps to the precipitating room. The zinc-boxes are from 24 to 36 in. wide, 10 to 12 in. deep, and about 20 ft. long.

Fine crushing is found to be unnecessary, as the ore is very porous, and much of it disintegrates into mud when solutions are applied.

It is interesting to note one change made in Mercur practice. Formerly the strong solution was run through the ore continuously, the surface being kept always covered. Now a series of washes is run through, the solution each time being drained down below the surface. The extraction has been increased thereby, and much time saved on each vat.

The solution used is from 0.1% to 0.3% in strength. It was at one time the practice to estimate the strength of the solution by its action in the zinc-boxes, and by its alkaline feel. At the present time more accurate methods are practiced. Still an instance has come within our notice of an operator determining the strength of his solutions wholly by their odor.

At the Mercur mill the practice was formerly to standardize solutions by adding cyanide to the lower end of the zinc-boxes, "the judgment of the operator determining the time and amount."

The zinc slimes were dried in an old retort belonging to the amalgamating mill. The door is closed, but not luted, and at about 160° C. the product ignites, producing fumes of a complex nature, causing salivation and headache. The slimes are finally taken from the retort, and the burning completed on a sheet-iron table. This product is then shipped to a smelter for refining.

At present the residues from a $12 ore assay about $1.75, giving an extraction of 85%. The average ore value is about $6 per ton. The cost of treatment is itemized as follows: Mining, 35c. per ton; railroad hauling and milling, 80c.; cyaniding the ore, $1.35—total cost per ton, $2.50. The consumption of cyanide is at present about ½ lb. per ton of ore.

In the Camp Floyd district there are nine cyanide mills, several

of which could not for some time be profitably worked by cyanide, on account of the presence of arsenic in the rock. Several more are in course of construction, however, with facilities for overcoming the arsenic trouble.

The Golden Gate Cyanide Works in this district, belonging to J. R. De La Mar, are the largest and newest works for the application of the cyanide process in the United States. It was the intention of the proprietor to make them an exemplification of the highest state of the art—an end which will probably be attained when certain serious mechanical difficulties, which were not foreseen by the projectors of these elaborate works, shall have been overcome. It is said that the results expected have not yet been attained, and consequently the metallurgical practice has not yet been definitely settled. These works, which were constructed in 1898, are built on a hillside with eight levels. In order to get the ore to the top of the works it has to be hoisted on an incline 800 ft. long. The mill is 294 ft. wide, and 420 ft. in length up and down the slope. The difference in elevation from top to bottom is 145 ft. The retaining walls, which are 2 ft. wide at the top and have a batter of 1 ft. in 12, required over 5,000 cu. yd. of rubble masonry. The various floors were constructed by blasting out the side hill. The broken stone thus obtained was used for the retaining walls and filling behind them. The mill is driven by power transmitted electrically a distance of 35 miles at a tension of 40,000 volts. The loss of energy in transmission is said to be only 5%.

At the works the 40,000 volt 3-phase current is transformed to one of 220 volts of 2-phase. The current is delivered at a contract price of $60 per h. p. The first section of the mill contains the coarse crushers, and in the second are the dryers. In the third section is the fine crushing machinery, which consists of four sets of 26-in. rolls and three sets of 36-in. Berthelet apparatus are used for sizing. There are six elevators with a lift of 60 ft. The fourth, fifth, and sixth sections contain the roasting furnaces, which are of Brown's straight-line design, four in number. Those intended for arsenical ores are estimated to have a daily capacity of 75 tons, while those for talcose ores are rated at 150 tons. The ore is stirred by the rabbles once each minute. One man attends to two furnaces. The gases are carried from the furnaces through 6 by 8 ft. flues into the main dust chamber, which connects with a steel stack 8 ft. in diameter and 85 ft. high, located on the hill above the buildings. The top of this stack is 275 ft. above the

lowest level of the building. The leaching department, which constitutes section 7, is 60 by 294 ft. It has two floors, the main floor supporting ten tanks 25 by 50 ft. and 5 ft. deep (presumably rectangular), and three solution tanks 20 ft. in diameter and 12 ft. deep. The tanks are supported by masonry piers. They are charged by hand from cars run on bridges over the tanks. The eighth section of the mill, which is 50 by 70 ft. and two stories in height, is for the precipitation department. It contains three precipitation tanks 14 ft. in diameter and 8 ft. deep. The tailings from the leaching tanks are discharged into cars which are run to the waste dumps. The building is constructed of steel.

Colorado.—In Colorado, within a few years, very extensive and excellent plants have been erected for the direct treatment of ores by the cyanide process. The details of the method of applying cyanide solutions in these works are not available; very little has been published on Colorado practice. An excellent, but brief, resumé of the field* has been written by Prof. Furman, of the Colorado School of Mines, which I take the liberty of quoting:

"The ores of the Cripple Creek district consist of porphyry (andesitic breccia), phonolite, decomposed granite, and quartz; and usually carry, on the surface, iron oxide, manganese oxide, and oxide of tellurium; below the water level the gold occurs in the minerals calaverite and sylvánite and is associated with more or less iron pyrites. The mineral fluorite frequently occurs in the gold-bearing veins. While the surface ores contain free gold they do not yield their gold contents by amalgamation, the gold usually being coated with oxide of tellurium, or some foreign substance, which interferes with its extraction by amalgamation.

"The extraction of gold from the surface ores, by potassium cyanide, presents no difficulties; but the treatment of the telluride ores without subjecting them to a preliminary roast has been attended with the drawbacks of extremely fine grinding and prolonged percolation in the vats (sometimes of 12 to 14 days in order to secure a fair extraction). At present all telluride ores are roasted 'dead' before leaching in the vats.

"The process as applied at the mill of the Brodie Reduction Co., situated about one and one-half miles south of the town of Cripple Creek, is typical of the method adopted for the treatment of these ores. The mill at present has a capacity of about 100 tons per day. The ore, as received from the different mines, is unloaded into

* *Mines and Minerals*, January, 1897.

bins, each lot being kept separate until it is sampled and paid for; the ore being purchased upon its value as determined by sample and assay, which is the invariable custom in the district. From the bins the ore is delivered, by hand, to a Gates crusher which reduces it to pieces not exceeding 1 in. in diameter. From the crusher it is elevated by a belt elevator, and passes through a Vezin sampler, which takes out a sample upon which the settlement as to the value of the lot is based. The crushed ore passes through the four-tube Argall dryer and thence to a Dodge crusher, thence it is elevated in a belt elevator to a revolving screen (iron wire 2-mesh), the oversize passing to Davis' rolls, whence it is returned to the screen, and the undersize passing to Krom rolls (14 by 20 in.). The product of the Krom rolls is elevated, divided, and delivered to four revolving screens (40-mesh brass wire cloth). The undersize is carried, by a screw conveyor, to the storage bins or roasting furnace, as is desired. The oversize passes to another set of Krom rolls whence it is elevated to the 40-mesh screens.

"The oxidized surface ores pass directly to the storage bins, while the unoxidized telluride ores pass to the roasting furnace. The furnace at these works is a Pearce Turrét, 40 ft. in diameter, with an annular hearth 8 ft. wide, the rabble arms being water-cooled.

"From the storage bins the ore is drawn into tram cars, each carload being weighed and dumped into the leaching vats. The vats are circular, and are constructed of No. 8 iron. Each vat is provided with a false bottom constructed of two layers of wooden slats covered with cocoa matting and No. 8 duck. The vats are provided with man-holes for sluicing off the tailings after the charge is leached.

"The stock solution is stored in steel tanks, covered with paraffine paint, and is kept at the proper strength by the addition of potassium cyanide. The solution is run in on the top of the ore and allowed to percolate. Two solutions are used, the strong solution containing from 0.5 to 0.75% potassium cyanide. The time of treatment varies from 70 to 100 hours, or more, according to the ore. After the gold is dissolved, wash-water is added to displace the cyanide solution. The strong solution is allowed to percolate for about 50 hours, when the weak solution is added, and afterward the wash-water.

"The solution, after passing through the filter, is conveyed by an iron pipe to the zinc-boxes for the precipitation of the gold.

There are two sets of zinc-boxes, one set for the strong and one for the weak solutions and wash-water. After passing through the zinc-boxes the solutions pass to their respective sumps, from whence they are pumped to their respective storage tanks. The zinc slimes are washed, treated, smelted, and the resulting gold bars shipped to the United States Mint at Denver."

The latest practice at the Brodie mill is to treat the ore raw with cyanide and concentrate the residues on Wilfley tables. These concentrates are roasted and re-leached. The residues from the second leaching are again concentrated, and subjected to a third leaching. Good results are claimed for this system, but we have no data of working results.

The largest and most representative cyanide works in Colorado are those of the Metallic Extraction Co., at Cyanide, a station on the Florence and Cripple Creek Railway, about 35 miles from Cripple Creek. The following account of cyanide practice at this plant is condensed from an article by Philip Argall, contributed to "The Mineral Industry," vol. vi., 1897:

At Cripple Creek the process has been successfully applied to the treatment of low-grade telluride ores, which proved refractory to ordinary mill processes, and were too low grade to stand the high smelting and transportation charges. Mr. Argall describes the ores of this district as "altered andesite, granite or phonolite, containing thinly disseminated iron pyrites and tellurium minerals, mostly calaverite, associated with quartz, and very often fluorite. At or near the surface, the tellurium is oxidized, and the gold, when visible, exists as an ocher-colored powder (mustard gold) or semi-coherent mass, very often retaining the form of the tellurium crystal from which it was liberated. The pyrite carries gold, but not to any great extent. Free gold also exists in the unaltered phonolite dikes, very often in paying quantities."

It was found that the oxidized ores could be easily leached; but that the unoxidized ores required roasting, which had the effect of setting the gold free, and rendering it amenable to the action of mill solutions. The crushing of the ore to suitable fineness for leaching purposes, and producing in the operation a uniform product, with the least possible percentage of dust, were desiderata of the first importance, and required much preliminary experimentation before work could be undertaken on a large scale. The demonstrated economical success of the cyanide process, as applied to this class of ore, led to the erection, by the Metallic Extraction

Co., of large works, which have been operated under the direct supervision of Mr. Argall.

The various steps in the operations carried on at these works are as follows: The ore is delivered at the sampling works from all parts of Cripple Creek; such ore being purchased on the basis of its assay value, less the treatment charges ($7 to $11 per ton), which includes the freight from mine to works. There are two sampling works. At No. 1 works the ore is crushed to about 1 in. cube by means of one No. 4 Gates crusher, and two No. 1 Gates crushers with intermediate screens. This ore is then cut down by means of a series of Vezin samplers and crushing rolls to a sample passing a 4-mesh, and containing only 2.5% of the original ore. This is further crushed and cut down until the final sample passes a 120-mesh screen. The main part of the ore is taken by special conveyors to the storage bins.

The No. 2 sampling works consists of a 12 by 20 in. Monarch crusher, and two 30 by 5 in. Blake Challenge crushers. Here, similarly, the ore is cut down and sampled, the bulk going to the storage bins. An admirable feature of this system is that the ore is handled but once in the crushing, sampling, and distribution to the storage bins. These bins have a capacity of 1,500 tons.

The next step is the drying of the ore, as a preliminary to screening. The dryer consists of four 18-in. steel, fireclay-lined tubes, nested together, and bound by two heavy tires. The tires rest on, and are driven by, steel-faced carrying rolls. The tubes are set on an incline, and revolve slowly, the ore being delivered at the upper end in four thin streams, which flow regularly and gently down the tubes, with the production of a minimum of dust. The ore is dried by the hot air and gases ascending the tube from the furnace, and is finally delivered to steel-bucket elevators, which carry it to the screens. The two dryers at the Metallic Extraction Co.'s works have an aggregate capacity of from 260 to 300 tons per day.

The ore, heated to about 300° F., is delivered to a series of screens and crushers, the system being as follows: The oversize from the coarse screen passes to crushing rolls, while the material passing the screen advances to the next finer screen. This is carried on until the desired fineness is obtained; the oversize passing each time to finer crushing rolls. The peripheral speed of the rolls is governed by the size of material; rolls receiving up to 1.25 in. cube, are run 350 to 400 ft. per minute; rolls receiving from 15 to 20-mesh for reduction to 40-mesh, 1,000 ft. per minute.

There are three fine-crushing mills at these works. From No. 1 mill, which reduces 100 tons per day to 40-mesh, the material passes directly to the leaching vats; from mills Nos. 2 and 3, which crush in the aggregate 370 tons per day to 30-mesh, the material goes to the roasters. Later on, with the increase in size of the roasting plant, it is proposed to send all material to the roasters, as the supply of oxidized material is growing very small and uncertain.

The crushing mills are situated adjacent to the roasters. Between the mills and roasters storage bins are provided for 800 tons of ore.

The tubular roaster is used. It is constructed on the same general principle as the dryer, except that it is much stronger, the tubes larger, and the discharge different. The tubes make one revolution in 4.8 minutes. They terminate in a hood, which is provided, along its periphery, with a series of holes, through which the ore escapes into a common hopper. These holes are covered with a band of iron, except immediately over the ore hopper, to prevent the escape of flames from the furnace. The furnace is a firebrick-lined steel shell mounted on wheels. The slow, steady advance of the streams of ore from the delivery end toward the furnace insures a gradual heat almost to the point of sintering, which appears to be necessary to drive off effectually the sulphur. If the sulphur is reduced from 2% to 0.1%, satisfactory extraction is obtained. The roasted ore is carried by conveyors traveling over water-cooled surfaces, to the leaching room.

The leaching tanks are built of steel, the largest being 50 ft. in diameter. The filters consist of cane matting supported by a wooden framework, and covered by 10-oz. duck. The residues are sluiced out by means of bottom discharge valves.

Zinc shavings and zinc dust are used for precipitating the gold. Mr. Argall is of the opinion that while zinc is in some respects unsatisfactory as a means of precipitation, it is the best available for the Colorado ores; and that electrical precipitation cannot compete with zinc as used at the Metallic Extraction Co.'s works.

With reference to the efficiency of old mill solutions as compared with new, Mr. Argall deduces the following important conclusions, after a series of exhaustive experiments: (1) Mill solutions will give equally as good extraction as pure solutions, but they require longer contact with the ore, say 10% longer; (2) mill solutions will give the same extraction as pure solutions, with about 25% less of cyanide; (3) the lower consumption is probably in part due to

the mixed cyanides in mill solutions being less sensitive to cyanicides, and partly to the potassium, sodium or other cyanides regenerated after the zinc is precipitated in the ore.

Consumption of Cyanide.—Mr. Argall finds that the quantity of cyanide present in solutions leaving a zinc-box is sometimes more, sometimes less, than upon entering; while the average shows no appreciable consumption of cyanide in the boxes.

Consumption of Zinc.—There is a consumption of 0.92 lb. zinc for each ounce of fine bullion recovered. Of this amount 0.4 lb. is dissolved in the boxes, and the balance reduced at the clean-up, on account of the fine state of division of the zinc, and its richness in gold.

It was observed that solutions rich in gold precipitated far more rapidly than weaker ones, other things being equal; and that where the solutions pass 1.5 oz. per ton in gold, the precipitate is yellow instead of black.

At the Metallic Extraction Co.'s works clean-ups are made weekly. The slimes and fine zinc are roasted to a dead heat, and then treated with hydrochloric acid, dried and smelted.

Montana.—One of the first cyanide plants of note erected in Montana was that at Revenue, Madison County, costing about $20,000. The first practice at these works was to collect the ore running from the batteries in settling pits, and to shovel this material into cyaniding vats for treatment. The overflow of slimes was reserved for future treatment. An extraction of from 80 to 87% was claimed. We have no information of recent practice at these works.

One of the largest plants in the United States is that recently erected by the Montana Mining Co., Ltd., at Marysville, for the treatment of accumulations of low-grade mill tailings. This plant has a capacity of about 400 tons per day.' I am indebted to Mr. C. W. Merrill, the superintendent, for the following information:

"The tailings beneficiated during the preceding summer (1897) by the new cyanide plant of the Montana Mining Co., the largest then in operation in this country, were taken from what is known as their No. 5 dam. This bed, the lowest of the five, situated in Silver Gulch, is about six miles below Marysville, and contains a much greater proportion of slimes than the upper beds, owing to the fact that it has been largely used for final settling and clarifying of the water. Therefore the first step in the process of treatment, that of putting the material in suitable shape for leach-

PLATE XII.

TAILINGS WORKS OF THE MONTANA MINING CO., LTD., MARYSVILLE, MONT.

ing, before loading it into the cars, is one requiring considerable care. In order that this point may be perfectly clear, it should be explained that in the operation of any lixiviation plant the material handled from day to day should be as nearly uniform in fineness as it is practicable to obtain it, and also that as few slime lumps as possible should be charged into the leaching vats, because they are not only impermeable to the solution, but soak it up, and do not release the value so dishonestly obtained. To accomplish this preparation, the bed is plowed, harrowed, and worked over continuously in order to dry and partially pulverize the lumps. The further precaution is observed of so selecting the material from different areas on the bed as to give comparatively uniform leaching charges each day.

"The second step, that of loading the cars, is accomplished in the old Mormon fashion, using traps or bridges over railroad cuts, and scrapers for moving the material. The pattern of the latter is known as the Fresno buck scraper, and the economy and advantage of their use will be appreciated by contractors when it is stated that we have been loading approximately 400 cu. yd. per day with six men and 12 horses, with an average haul of not less than 100 ft. The use of the steam shovel in this work is prohibited by the fact that the material should be dried and pulverized on the surface of the bed before loading.

"The cars used are of 3 tons capacity with bottom discharge, and 16 loaded cars make the train for a 22-ton locomotive. The latter is of the saddle-back type, 9-ft. wheel-base, with six drivers carrying all the weight. The rails are 56-lb. Northern Pacific third class, the gauge 3 ft., the maximum curve 30°, and the maximum grade 3.5%.

"The plant is 2½ miles above the No. 5 dam, and the tailings, after leaving the cars, pass in an almost continuous stream to the sheet-iron lining of the bin, and out the gate to a 24-in. 4-ply belt conveyor, which conducts them to a revolving chute or distributor, and this in time fills a vat 38 ft. in diameter by 9 ft. deep with 400 tons of tailings, in about 8 hours. The great advantage in filling a tank in this way is that it gives a charge of more uniform permeability than any other method of filling known to the writer.

"There are four of these tailings vats, each with its bin, conveyor, and distributor, and one is charged daily, thus giving four days to complete the treatment of each charge, which consists in saturation, lixiviation, washing, and discharging. The latter is

accomplished by sluicing with two 2½-in. hose, the water being under a 60-ft. head, through four side discharge gates, and one bottom discharge valve in the center of the vat. By this method 400 tons are discharged in three hours or less, at a total cost of less than 2c. per ton.

"The solution tanks are six in number, and consist of four precipitating tanks 22 ft. in diameter by 14 ft. 9 in. deep, and two storage or supply tanks, 38 ft. in diameter by 9 ft. deep. There are also two water tanks 22 ft. by 14 ft. 9 in. for the storage of 80,000 gal. of water.

"The power equipment consists of a 50 horse-power boiler, one 28 horse-power engine, one 10 horse-power engine for running conveyors, one 30-light dynamo, one Knowles economical geared pump, with 4-in. inlet and discharge, two Worthington all-iron duplex solution pumps, with 5-in. suction and 4-in. discharge, and one 2-drill air compressor.

"The tailings are the lowest-grade in gold of any being worked in this country; they are the most rebellious of any worked in the world, because they contain copper carbonates and sulphide, tetrahedrite, arsenical polybasite, and ruby silver. The plant operates under the most unfavorable climatic conditions, being in the most northern latitude of any known to the writer; but in October and November it exceeded the prediction of profit by nearly 50%, and netted about double that which the company thought would justify the erection of a plant."

Nevada.—One of the largest plants in the West is that at the De La Mar mine in Lincoln County, Nev. Ore is treated direct by a modification of the cyanide process known as the Kendall process, in which peroxide of sodium is added to the working solutions. A very high extraction is claimed at these works. Zinc fume has been extensively experimented on as a precipitant, and is now used in practice with alleged success. These works are said to have produced $1,700,000 in gold in 1897.

At Candelaria, a plant has been for some time in successful operation on silver tailings.

At Bellville, it is said that a modification of the cyanide process is about to be applied to the immense accumulations of silver tailings in that vicinity.

The Eureka Cyanide Co. has recently erected a very complete, modern plant on the Carson River for the treatment of tailings from the ores of the famous Comstock lode. For the following

description of these excellent works, and the accompanying photographs, I am indebted to Mr. A. J. McCone, one of the owners: This plant was constructed in 1897, and extensively enlarged and improved in 1898. It is located on the Carson River, about 10 miles from Virginia City, and on the site of the Eureka quartz mill. This mill was destroyed by fire in 1892 after a six-years' run on Consolidated California and Virginia ores, the battery slimes from which were carefully conserved in reservoirs. The sands were separated from the pan-tailings by means of Lyman riffle sluices, to be mixed subsequently with the slimes to form a product which could in turn be treated with facility by the Park amalgamating process, then so largely employed in working Comstock slimes and tailings. In 1893 a 12-pan tailings mill was built on the site of the Eureka mill, and was operated for four months on the combined sand and slime deposit; but the slump in silver compelled the owners to discontinue operations. In 1894 the Peck brothers, with a process of their own, concentrated the values from the slime deposit, leaving the sands untouched, the latter not being suited to the machinery employed. In 1897 the projectors of the present Eureka cyanide works secured this deposit of sands, and a lease of the site and improvements. They also acquired possession of deposits of slimes in Virginia City and at the Brunswick mill on the Carson River, and deposits of sand and slimes at the Morgan and Mexican mills at Empire City, aggregating 200,000 tons. These several deposits, while separated several miles, one from the other, are all located close to the line of the Virginia and Truckee Railroad. This road delivers the material from the various deposits into bins having a capacity of 800 tons, built under a spur trestlework projecting from the main line of track. This spur is about 2 miles from the Eureka plant, and is on the hillside 200 ft. above the Carson River bottom. The cyanide company unloads the cars of the railroad into the upper bins, and by means of a steep gravity tramway lowers the slimes and tailings into other bins located convenient to its own line of railroad at the foot of the hill. From the lower bins the company hauls the material to the plant two miles away. The railroad used for this purpose is 30-in. gauge, built with 16-lb. steel rails, and is equipped with bottom discharge, hopper-shaped cars of about 3 tons capacity, and a 5½-ton locomotive.

On arriving at the plant the tailings are discharged into a bin, arranged in four compartments to accommodate products of vary-

ing fineness. This bin, having a capacity of 150 tons, is located under the track. From the four discharge hoppers the different products are automatically fed into a conveyor in the quantity required. In their passage to the plant they are mixed, by means of a branch conveyor, with the required quantity of coarse sand from the Eureka deposit, thus forming a uniform product suitable for percolation. As the sand passes along it receives the requisite quantity of lime from an automatically discharging hopper. Finally it is dropped into the boot of an elevator, is lifted 47 ft. into the ventilator of the tank house, and is discharged by means of chutes into the treatment vats.

This cheaply run, automatic conveying system grew out of the necessity for handling tailings as economically as possible in these works, the material being derived from the more recently exploited, low-grade ores of the Comstock lode. Three-fifths of the value of the tailings is in silver, and they do not work up to the high percentage obtained in the gold districts. In order to make this material pay, a plant as automatic as possible in its operation had to be designed, and it is safe to say that the designers have secured quite a perfect plant in this respect. The tailings are conveyed quickly and cheaply from the main railroad line to the plant, as the bins and chutes are all made to work by gravity, and are arranged to clear themselves well. The conveyor under the receiving bin at the plant, with its bar-feeding mechanism, handles the material cheaply and perfectly. The branch conveyor running off at right angles to the main conveyor into the sand pile near the plant has a platform and bin with automatic feed connections at its base, and carries into the main conveyor enough sand to make up the percolating mixture. The elevator is a slow-speed perfect discharge chain-elevator with steel-bushed chain and large buckets, and discharges into the conveyor on the upper floor from each bucket the exact quantity delivered at a stroke by the lower conveyor. The upper conveyor is located high enough above the vats to admit of placing 45° chutes under the conveyor-box, which deliver the material 3 ft. 6 in. above the vats. There are three chutes to each vat, which fill the vat (150 tons) in about four hours. The sand must be hand-distributed, but the conveyor-attendant does this work with little trouble.

The conveyor is a reciprocating machine with pendant, hinged flights fixed on a horizontal tubular bar, the whole operating in a narrow wooden box. These flights are rigid on the forward motion

of the bar, each pushing the sand ahead toward its destination a distance equal to the stroke of the bar; returning, each flight-blade moving on a hinge, rides over the back mound of sand, and advances it in turn toward its destination.

There are eight tailings vats, 24 ft. inside diameter, and 7 ft. 10 in. deep above the filter, arranged in two rows. They are provided with the usual solution, water, and drainage pipes. Each vat is provided with a separate drainage pipe to the zinc-boxes, the conditions of location having made it impossible to set up intermediate solution tanks. Each vat receives weak and strong solution through a single main, so designed that only 10 valves control the solutions to the eight vats.

There are six 10-compartment zinc extractor boxes made of wood. Each compartment is 13 by 24 in. and 20 in. deep, with bottoms sloping toward a side launder for conducting away the slimes at the clean-up.

The solutions, after passing through the zinc-boxes, fall into sump tanks. There are two of these tanks, one for strong and one for weak solution, each 25 ft. in diameter and 8 ft. deep. By means of rotary pumps the solution from the sumps is elevated to the storage tanks, located about 15 ft. above the top of the sand vats, in a separate building. These tanks are 16 ft. in diameter and 12 ft. deep. The cyanide is stored in the tank room, and the solutions are here brought up to proper strength and run through $2\frac{1}{2}$-in. service mains to the sand vats. Two gold-solution tanks have been installed for a special purpose. At times a pump is connected with the drain from under the filter-cloths of the sand-vats, and by means of suction percolation is accelerated. This pump discharges into the weak or strong gold tank, whose contents are drawn off to the zinc-boxes whenever convenient.

At the end of the clean-up launders, conducting from the zinc-boxes, are two cast-iron filter-boxes of strong construction, which partially dry the slimes before they are submitted to acid treatment.

The acid treatment agitator is a wooden tub, 6 ft. in diameter and 30 in. deep, provided with stirring arms affixed to an iron spider, driven by bevel gear and pinion at a slow rate of speed. The zinc-slimes are treated with dilute sulphuric acid until the zinc is completely dissolved; by agitating with frequent charges of water, the residues are "sweetened," or washed free from acid.

Then comes a treatment that is quite unusual in cyanide work, recovery of quicksilver. The tailings handled, being from the

Comstock lode, have passed through the Washoe process of treatment, that is, crushing, settling in tanks, grinding in pans, preparing for amalgamation with sulphate of copper and salt, amalgamating with mercury, and settling in slow-motion settlers to gather the resulting amalgam. The loss of quicksilver in this process as applied to Comstock ores has always been heavy. The quicksilver appears in the tailings finely divided as a chloride of mercury. A percentage of this is dissolved by the cyanide solutions, precipitated on the zinc, and goes through the various stages of cleaning, filtering, and acid treatment with the slimes. It was seen from the beginning of operations that this mercury might be made an important addition to the output of the plant, and retorting was resorted to in order to separate the mercury from the slimes, as well as to dry the slimes. This is accomplished in a "boat" in an ordinary mill retort. The vapor from the retort, after passing through the usual condenser, is directed through a hydraulic seal in a vacuum-chamber, an improvement on the ordinary retorting method of much apparent value.

The dried slimes, after coming from the retorts, are properly fluxed, melted into bars, and sent to the mint in Carson City.

The plant is well equipped with shops, tools, and fixtures. A large quantity of zinc is consumed, and lathes are kept constantly running cutting the shavings. The machinery of the plant is operated by water power from the Carson River, but steam-engines are in place to furnish power during the dry season.

The assay office is perfectly appointed. A testing plant with 150-lb. capacity is established, and tests are made from several vats each month, modifications of the process being made as the results demand.

The quantity of material put through each day is 150 tons, equal to the contents of one vat. Each vat gets 7½ days' treatment, and seems to require it to extract a satisfactory percentage.

The vats are sluiced out by streams under 75 lb. pressure, the discharge-gates being in the sides of the vats, a penstock from each gate connecting with an ample V-shaped launder. The launders from all the vats discharge into a flume, the bottom of which is covered with grate-like riffles, where some quicksilver and amalgam are caught.

California.—At Bodie about 8,000 tons of tailings per month are being treated at the various plants. At Lundy, 25 miles distant, a 75-ton plant has commenced operations on a 50,000-ton

accumulation of old tailings which were originally run into Lundy Lake, where they lie submerged. They are pumped directly from the lake into the cyaniding vats. Smaller tailings plants are scattered throughout the State, but the application of the process has not been very extensive. The extraction of gold by ordinary stamp-mill processes has proved eminently satisfactory, and the mill tailings, especially in the mining centers of the northern counties, have, in most instances, been too low-grade for cyaniding.

One of the most interesting cyanide works in the West is a small plant erected by Dr. Scheidel at Angels Camp, Calaveras County, in 1894. It is a model of construction in its way, and in its operation proved to be a noteworthy success. It was designed for the treatment, by agitation, of an almost impermeable class of concentrate slimes. The following description of these works is taken from Dr. Scheidel's work on the "Cyanide Process:"

"This plant is completely built of steel and iron, and consists of the following parts: A vertical cylindrical agitator, 5 ft. in diameter by 5 ft. high, of $\frac{1}{4}$-in. steel plate, with a cast-iron bottom 2 in. thick, with strengthening ribs underneath; to the bottom is cast a cone, through which passes the vertical shaft, which carries four arms. To these are attached the four paddles of $\frac{1}{8}$-in. steel, 6 in. wide, twisted like the blades of a propeller. A ring connecting the four paddle arms gives greater stability to them. The shaft with the paddles can be raised, by means of a screw-spindle, 4 ft. above the bottom of the apparatus. A wrought-iron ring, 3 in. wide and $\frac{1}{2}$ in. thick, riveted outside around the top of the agitator, strengthens the structure. The driving gear is placed below. An opening 4 in. diameter, in the bottom, discharges tne contents of the agitator through a pipe, furnished with a stopcock, on to Scheidel's patent vacuum filter, placed on the floor below and in front of it. Here a perfect separation of the cyanide gold solution is effected from the residues. This filter is built of $\frac{1}{4}$-in. steel, with bottom $\frac{3}{8}$ in. thick; it forms a rectangular box 3 ft. 6 in. deep, 7 ft. long by 5 ft. wide; 2 ft. above the bottom is a perforated steel filter-bottom of $\frac{3}{8}$-in. boiler plate, made in three movable sections, supported by angle-iron running around the sides, and by the vertical support of double T-iron. The perforations are of $\frac{1}{2}$-in. diameter, at a distance of $\frac{1}{2}$ in. from each other. This filter-bottom fits closely to the sides of the apparatus; it is covered with a blanket, which is kept in position by bars running along the four sides, and fastened by thumbscrews. A grating in three sections

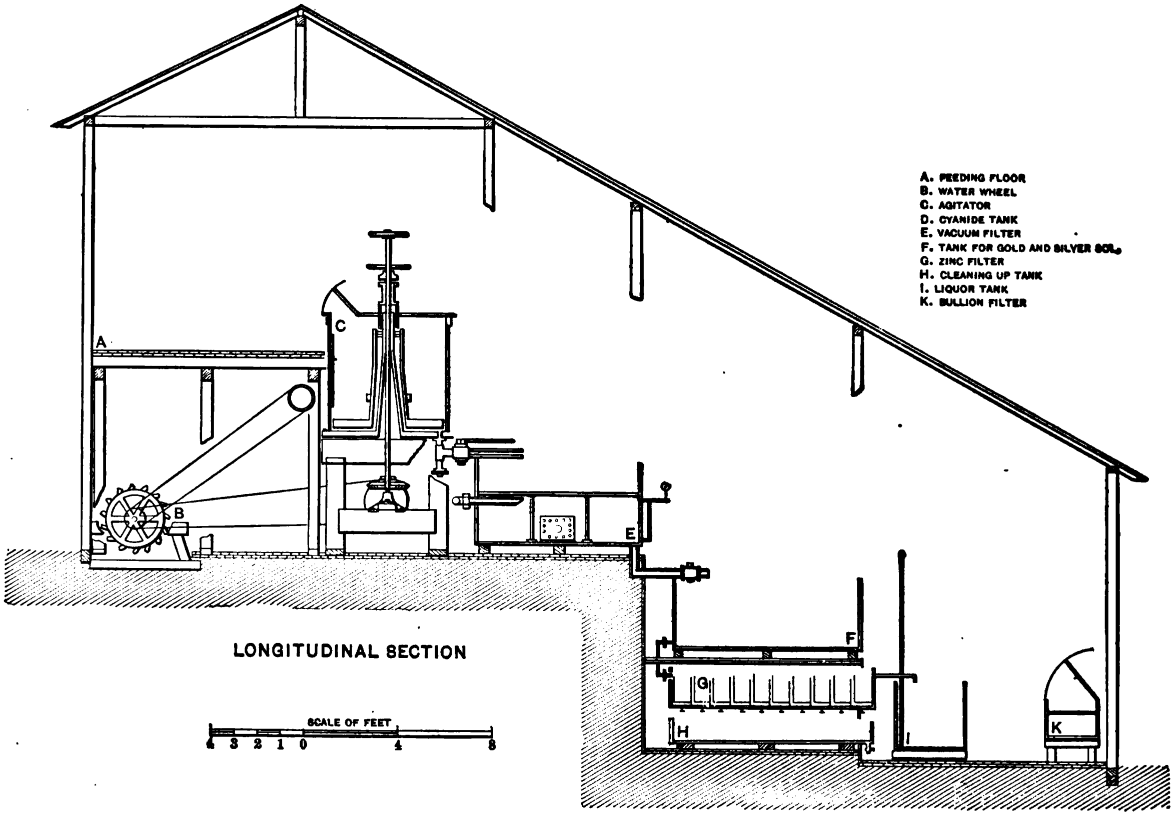


FIG. 19.—SCHEIDEL'S AGITATION PLANT FOR THE TREATMENT OF SLIMES AT ANGELS CAMP, CAL.

of ⅝-in. round iron serves to protect the cloth; the intervals of 3 in. between the bars are filled in with coarse sand. The filter partition divides the apparatus into two compartments, one above the other; the lower forms a closed box, which is in connection with a duplex vacuum pump, by means of which the air can be rarefied when the filter-bottom is covered with pulp. The upper part, above the filter-bottom, receives the contents of the agitator. The real bottom of the apparatus has a discharge with a 3-in. stopcock, for running off the filtered solution into either one or the other of the two solution tanks, which are standing on the floor one step lower, in front of the filter. All cocks and taps of the plant are of considerable diameter, which secures a quick charge and discharge. The filter is provided with a gauge to indicate the height of the solution within, a gauge indicating the degree of vacuum, an air-tap to permit influx of air when the filtered solution is being discharged, and a manhole.

"The mode of working the plant is this: The cyanide solution is charged into the agitator, the paddles are set in motion by revolving the shaft, the ore is charged by degrees, and the agitation is kept up for the required time, after which the pulp is discharged from the agitator on to the filter. The vacuum pump is then set in motion, and filtration under the influence of atmospheric pressure will at once commence. The solution will soon be sucked through; then washing follows, first with liquor from former operations, which has already passed through the zinc-boxes, and finally with clear water. These operations of filtering and washing take about two hours. It is advisable to suck the tailings as dry as possible before each new wash is put on, which permits the complete removal of the gold solution with a very small amount of liquid, one-half ton of which is sufficient for washing a charge of two tons of ore. If the filtration has been properly managed, no degree of continued washing can improve the results. The filtered solutions are clear. The first, or original solution, together with the first wash, will be run off into one of the two solution tanks in front below the vacuum filter; the following washes run into the other. These tanks are 8 ft. long by 3 ft. wide and 3 ft. deep, made of ¼-in. steel.

"Each of the tanks is in connection with a zinc precipitation box, 9 ft. long by 21 in. deep and 9 in. wide, divided into ten compartments; there is an interval of 1 in. between each two compartments. (See Fig. 10.) The false perforated steel bottoms of the

chambers, which can be removed if desired, are 2½ in. above the true bottom of the box. The bottom of the box has a number of 1-in. iron faucets, one corresponding with the center of each filter compartment; the sides of the box are 4 in. higher than the partitions within, which insures absolute safety against the liquid running over the sides of the box, if one or the other compartments should become blocked. The gold solution flows into the box through a 1-in. cock, enters the first compartment from below through the perforated false bottom, percolates the zinc shavings placed thereupon, leaves it, and enters the second, and so forth. A steel settling tank, 12 in. deep, 12 in. wide, and 9 ft. 3 in. long, is placed below the precipitating box for receiving the bullion when cleaning up. The zinc used consists in turnings $\frac{1}{3000}$ in. thick, turned from cast zinc cylinders on a lathe; 2 lb. fill one compartment. The solution passes through the box at the rate of 700 gal. in 24 hours. The bullion precipitation of the solution is very efficient as it passes from compartment to compartment, which amounts to passing ten times through a zinc column 14 in. high by 9 in. square, as shown by the following table of analysis:

One Ton of Liquid Contains—	Gold.			Silver.		
	Oz.	Dwt.	Gr.	Oz.	Dwt.	Gr.
Originally	5	14	0	2	4	8
After 14 in. of zinc column		16	1		5	3
After 28 in. of zinc column		5	9		1	10
After 42 in. of zinc column		2	14			17
After 56 in. of zinc column		1	4			12
After 70 in. of zinc column		1	1			11
After 84 in. of zinc column			22			3
After 98 in. of zinc column			15.43			0
After 112 in. of zinc column			14.96			0
After 126 in. of zinc column			13.36			0
After 140 in. of zinc column			12.34			0

"The solution leaving the zinc-box contains only 12.34 grains of gold, or 0.0045% of its original contents. Simultaneously the solutions were analyzed for available cyanide, but no decrease in the strength of the solution, which remained constantly at 0.3185%, could be ascertained. At another period I studied the solubility of zinc in cyanide solution, of which I give the following figures: 0.2634 grams of filiform zinc were submerged in 50 c.c. cyanide solution of 0.26%; after seven days of frequent agitation these were reduced to 0.2584 grams, and after 56 days to 0.2252 grams, which means that after seven days 1.98%, and after 56 days 14.47%, of the

zinc was dissolved. From this observation it follows that the loss of cyanide in the precipitating boxes, by means of its being taken up by zinc, is sometimes overestimated. The washes pass through a similar precipitating box. The liquids, when leaving these boxes, go as liquor No. 1 and liquor No. 2, into tanks of the same size as the solution tanks, from whence a pump will deliver them wherever wanted. Liquor No. 1 serves for making up the new solution for the next charge; liquor No. 2 is used for washing purposes on the vacuum filter. No liquor ever leaves the works; the quantity in circulation remains stationary.

"The bullion obtained from the zinc-boxes is passed through a sieve to the bullion vacuum, which itself is a miniature reproduction of the vacuum filter. It has the following dimensions: Length, 2½ ft.; width, 2 ft.; total depth, 1 ft. 6 in. The perforated filter-bottom is fixed 12 in. above the true bottom. The bullion is very slimy; in fact, it is the more slimy the freer it is from zinc. Its filtration and washing take some time. When the mass is tolerably dry, it is put into a wooden tub and treated with diluted sulphuric acid.

"The steel of the tanks has not as yet shown any effects from cyanide, nor does it exercise any influence on the solutions. For agitation the material requires an amount of solution equal to 30% of its weight, and six hours of time.

"The average consumption of cyanide, calculated from a large tonnage of slimes treated, amounted to 4.3 lb. per ton, costing \$2.27; the labor amounts to \$1; the total expenses of treatment by cyanide to \$3.50 per ton. The average extraction amounts to 93.18% of the gold, and 90% of the silver assay-value; as high as 96.57% of the gold has been extracted in some instances. The extraction of the gold during the agitation goes on as follows, the percentage of extraction following in parentheses the amount obtained per ton: Sample before treatment, \$88.00; after 1 hour's agitation, \$13.00 (85.23); after 2 hours', \$11.00 (87.50); after 3 hours', \$7.00 (92.05); after 4 hours', \$7.00 (92.05); after 5 hours', \$6.00 (93.18); after 6 hours', \$5.00 (94.31); after 7 hours', \$5.00 (94.31); after 8 hours', \$5.00 (94.31). Within the first hour 85.23% of the gold is extracted; during the following five hours the increase of extraction is slow and irregular; after six hours no further extraction takes place. The treatment of the slime concentrates by agitation was preferred on account of its quicker, cheaper, and better results, as compared with percolation. Other sulphurets, such as the Frue

vanner concentrates of the Utica, Madison, and Eureka mines, were treated on a more or less extensive scale by the same plant; results were, however, not very satisfactory on account of their coarseness. All sulphurets of the Utica mine are pure sulphide of iron."

Cyanide plants are being successfully operated in Arizona, South Dakota, Idaho, Oregon, and Lower California.

In Arizona the Congress Gold Mining Co. and the Harquahala Gold Mining Co. have large works for the treatment of tailings. At the Planet-Saturn mill, in Fool's Gulch, the ore is being roasted previous to cyaniding, a procedure which has been introduced with such excellent results in Colorado and Utah, and is likely to be an important adjunct to cyanide practice in the future.

In South Dakota large works for the direct treatment of ore are in operation at Deadwood. At De Lamar, Idaho, cyanide lixiviation has recently superseded the Pelatan-Clerici process. Ore at the North Pole mill, in Baker County, Ore., is first roasted in a Brückner furnace, previous to cyaniding.

In Lower California a small plant is being operated by the Ybarra Gold Mining Co. for the treatment of a deposit of mill tailings.

CHAPTER XVI.

CONCLUSION.

AN attempt has been made, in the foregoing pages, to describe an elementary system of cyanide practice, based upon the fundamental principles of the process. At the same time, the great variations in mill practice in different parts of the world have been insisted upon and illustrated. These variations, resulting as they do from the solution of a variety of metallurgical and mechanical problems, require no commentary to make it evident to the reader that cyanidation, while not so complicated as some other processes for gold extraction, is by no means the simple affair it is popularly thought to be. "It has been stated," says Mr. Janin,* "that the cyanide process is so simple that the plant may consist of old barrels or tanks, placed together in any way, with crushing machinery of any description. The absurdity of such a statement is apparent to the metallurgist, but not always to the miner. The process is far from being simple; and the proper arrangement of the plant not only increases the percentage of extraction, but reduces working expenses. With low-grade ores, financial success is often due to the perfect mechanical arrangement of the metallurgical plant." There is probably no mining process in which the perfection of minute mechanical details means so much from an economical standpoint, or one which, metallurgically considered, is susceptible of so many variations. This is, of course, a natural consequence of the great end aimed at by a process adapted to working low-grade material; namely, the obtaining of the highest possible efficiency, at the least possible cost.

It is gratifying to reflect that the need of a close study of details, and of a proper conception of the imperfections of the process, as at present understood, are fully appreciated by operators in the best conducted plants. Yet it is to be feared that lax and unscientific methods prevail in many mills, a result, perhaps, of too much

* "The Mineral Industry," vol. i., "The Cyanide Process."

prosperity, or of the fatuous self-complacency of not over-intelligent operators.

The fact has been emphasized in these pages that the process has by no means a wide application. We have seen how many circumstances may interfere with its success; and how essential it is that each ore should be thoroughly investigated on its own merits. Some of the more important problems connected with the process have been ingeniously solved. The preliminary roasting of ores as a means of driving off noxious substances which interfere with cyanidation, has in a measure revolutionized mining in Colorado. It seems probable that further investigation will reveal other methods of preparing rebellious material, which will in time extend the application of the process to more common use.

The rapid progress made in the art of cyaniding during the publication of this work has caused me to modify or expand, in the last chapters, certain conclusions reached in earlier pages. This was done only when it was found too late to make alterations in the text itself. I refer particularly to the roasting and cyanidation of concentrates, the application of zinc-fume as a precipitant, and to certain data of cost of treatment in South Africa and elsewhere.

I anticipate the criticism that in these inadequate notes my deductions have been mainly drawn from an experience in Bodie practice. It will be objected that Bodie practice is limited, and perhaps far from the best; and that a system based upon a practice neither varied nor typical, is necessarily narrow. I can only revert to an admission already freely made, that the process, in its practical application, is susceptible of endless variation; and that Bodie practice differs quite as much from Colorado practice as the latter does from that of South Africa. It seemed perfectly justifiable, in view of the rapidly progressive and unsettled state of the metallurgy of cyanidation, to accept the Bodie system as the simplest possible illustration of the principles upon which the process is based.

INDEX.

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TABLE OF CONTENTS.

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